Digitized by the Internet Archive
in 2010 with funding from
University of Toronto

http://www.archive.org/details/transactions20cana

(13)

FEB 23 1918

THE TRANSACTIONS

OF THE

Canadian Mining Institute

1917

EDITED BY THE SECRETARY

VOLUME XX

◈

"The Institute as a body shall not be responsible for the statements and opinions advanced in the papers which may be read or in the discussions which may take place at its meetings."—*By-Laws, Par 48.*

◈

350819 38. 5. 20.

PUBLISHED BY AUTHORITY OF THE COUNCIL AT THE SECRETARY'S OFFICE
503, DRUMMOND BUILDING, MONTREAL, QUE.

OFFICERS AND COUNCIL, 1917

Past Presidents

Dr. Frank D. Adams	McGill University, Montreal, Que.
Mr. G. G. S. Lindsey	Bank of Toronto Bldg., Toronto, Ont.
Dr. W. G. Miller	Bureau of Mines, Toronto, Ont.

President

Mr. Arthur A. Cole	T. & N. O. Ry., Cobalt, Ont.

Vice-Presidents

Mr. D. B. Dowling	Geological Survey, Ottawa, Ont.
Mr. Chas. Ferfie	Intercolonial Coal Mining Co., Limited, 143 Dominion Express Bldg., Montreal, Que.
Mr. Thos. W. Gibson	Bureau of Mines, Toronto, Ont.
Mr. M. E. Purcell	Con. M. & S. Co. of Canada, Rossland, B.C.

Councillors

Mr. Leland D. Adams	Weedon Mining Co., Ltd., Room 605, Eastern Townships Bank Bldg., Montreal, Que.
Mr. Manly B. Baker	School of Mining, Kingston, Ont.
Mr. E. E. Campbell	Granby Con. M.P. & S. Co., Anyox, B.C.
Mr. W. A. Davidson	School of Technology, Calgary, Alta.
Mr. Theo. C. Denis	Supt. of Mines, Quebec, Que.
Mr. W. J. Dick	Commission of Conservation, Ottawa, Ont
Mr. Norman R. Fisher	Haileybury, Ont.
Mr. Thos. Graham	Canadian Collieries, Ltd., Cumberland, B.C.
Mr. E. Reginald Hore	263-5 Adelaide St. West, Toronto, Ont.
Mr. Gomer P. Jones	Daly Reduction Co. Limited, Hedley, B.C.
Mr. G. C. Mackenzie	Mines Branch, Dept. of Mines, Ottawa, Ont.
Mr. W. F. McNeill	912 Herald Bldg., Calgary, Alta.
Mr. Edward P. Mathewson	507 Royal Bank Bldg., Toronto, Ont.
Mr. N. C. Pitcher	13101-103rd Ave., Edmonton, Alta.
Mr. W. E. Segsworth	103 Bay St., Toronto, Ont.
Mr. F. H. Sexton	Nova Scotia Technical College, Halifax, N.S.
Mr. Clifford E. Smith	National Club, Toronto, Ont.
Dr. Alfred Stansfield	McGill University, Montreal, Que.
Mr. Jos. H. Stovel	Box 688, Sudbury, Ont.
Mr. M. W. Summerhayes	Porcupine Crown Mines, Timmins, Ont.

CONTENTS

PAGE

Presidential Address, by Arthur A. Cole ... 1

Sodium Sulphide Precipitation of Silver at the Nipissing Mine, by R. B. Watson ... 6

Milling Practice at the Buffalo Mines, Cobalt, Ont., by Robert E. Dye ... 11

Present Flotation Practice on Cobalt Ores, by J. M. Callow and E. B. Thornhill ... 28

Canadian Wood Oils for Ore Flotation, by C. S. Parsons and R. E. Gilmore ... 38

Notes on Milling Practice at the McIntyre Mine, Porcupine, Ont., by A. Dorfman ... 93

Notes Concerning Local Treatment for Certain West Kootenay Silver Ores, by Francis A. Thomson ... 110

The Concentration and Marketing of Canadian Molybdenite, by H. H. Claudet ... 121

Electro-Deposition of Zinc from Aqueous Solutions, by E. P. Mathewson. ... 135

Electrolytic Zinc in Eastern Canada, by E. E. Watts ... 161

The Formation of Zinc Ferrate, by E. H. Hamilton, G. Murray and D. McIntosh ... 168

The Kingdon Lead Mine, by John E. Hardman ... 180

Further Notes on Yukon Mining Problems, by Henry M. Payne ... 188

Countercurrent Decantation, by L. B. Eames ... 207

Refractory Materials in Canada, by N. B. Davis ... 226

The Development of Canadian Magnesite, by Harold J. Roast ... 237

The Partridge Smelting Furnace, by Allen R. Partridge ... 256

The Occurrence and Testing of Foundry Moulding Sands, by L. Heber Cole ... 265

Potash, Its Present and other Possible Sources, by C. W. Drury ... 292

Mining Methods at the Magpie Iron Mine, by A. Hasselbring ... 321

The Present Position and Future of the Iron and Steel Industries in Canada, by Corbett F. Whitton ... 333

Opportunities for the Establishment of an Iron and Steel Industry in British Columbia, by Robert R. Hedley ... 389

Methods of Mining in Crowsnest Pass District, Alberta, by Raoul Green. ... 400

Lubrication of Mine Machinery, by James A. Boyd ... 409

Pulverized Fuel for Locomotives, by J. S. Coffin, Jr. ... 416

The Future of the Coal Trade of the Empire, by Allan Greenwell ... 434

Bituminous Sands of Northern Alberta, by S. C. Ells ... 447

Notes on the Electro-Chemical and Metallurgical Possibilities in Canada, by H. E. Howe ... 460

Magnesium, by J. C. King ... 471

Some Suggestions on Filing and Indexing, by Frank E. Lathe ... 481

A Plea for Definite Training for Social Responsibilities by Means of our Educational Institutions, by C. V. Corless ... 491

PRESIDENTIAL ADDRESS

By Arthur A. Cole

Annual Meeting, Montreal, March, 1917.

Conditions, in general, during the past twelve months in respect of the mining and metallurgical industries of the Dominion have been satisfactory. The abnormally high prices of metals have increased profits and stimulated output. Even silver, the market for which during the first eighteen months of the war was depressed—in fact, for a time the market quotation for this metal was the lowest ever recorded—has since appreciated notably in value, and now commands a higher price than at any period during the past twenty years. The result of these favourable conditions is indicated clearly in the returns of the mineral production of Canada for the year just closed, representing as they do a new high level mark. The year was also a busy one for the Canadian Mining Institute, among whose activities may be mentioned, in particular, an important work, initiated at the request of the Rt. Hon. Sir George E. Foster, the purpose of which was to ascertain what practical measures might be taken to ensure that the control of the mining industries and metal trades of Canada be secured and retained by Canada. The services of the Institute were utilized also by the Department of Militia, to which assistance was rendered in respect of the selection and appointment of mining engineers applying for officers' commissions with tunnelling units; as well as in the recruiting of miners. Furthermore, in response to recommendations originating from the Institute, an investigation was undertaken under Federal Government auspices by the Forest Products Laboratories and the Mines Branch, to ascertain the feasibility of the production of flotation oils from Canadian wood products. Important results from this investigation will, I understand, be submitted to you very shortly. It should be mentioned also that the influence of the Institute was exerted successfully to avert a threatened restriction of the supply of cyanide to Canadian mines from Great Britain. The action that was taken, and the representations that were made, to prevent the creation of a condition that would have had a most serious and numbing effect on our precious metal industries, were not only successful

in achieving their direct object, but were conducive to the bringing about of the present satisfactory situation whereby Canada is assured of receiving, in future, ample supplies of cyanide at a lower price than anywhere else in the world.

During the year also, a metallurgical section of the Institute, was successfully organized, and already has afforded us an opportunity of extending the scope of our endeavour and usefulness. In short, the past year has been one of the most active as well as the most successful in the history of the Institute. This is due mainly to the loyalty of and the interest displayed by our officers and members. In particular, I desire to express grateful acknowledgment of the services rendered by the members of the Council, by those who served on special committees during the year, and by the executive committees and officers of our local branches.

The activities I have enumerated constitute a part merely of the work undertaken by the Institute in the general interests of the mining and metallurgical industries of the Dominion, of which it is now recognized as the mouthpiece, and as such its influence is steadily growing. Our membership is also increasing, but possibly at not so rapid a rate as it might. This may be explained, I think, mainly by the fact that much of the capital invested in the development and operation of mines in this country is of foreign origin. In a great number of instances the members of the technical staffs of these mines are not natives of Canada, and have little, if any knowledge of the work that is being done and has been done by the Institute to benefit the industry. Mining men, as a class, however, in common with all who travel extensively, are distinguished by their lack of insularity, and their breadth of vision. They are also fair-minded; and consequently I am convinced that our membership might be increased materially if only representations concerning the aims and accomplishments of the Institute were intelligently made to all those engaged in mining in Canada (no matter of what nationality) who are eligible to become members, but have not yet applied for membership. Hence, I suggest that a strong and energetic central committee be formed to consider and prosecute this project. By enlisting the co-operation of the local branches much of the territory could be covered effectively;

but the success of any efforts that can be made in this direction must necessarily depend in a large measure on the zealous co-operation of individual members, whose assistance therefore towards the attainment of the aim to make the Institute an even more representative organization than it now is, is earnestly invited.

Special attention, moreover, should be directed to enlisting the co-operation of the large metallurgical and mining companies throughout the country. Some of these are well represented in our Institute, others, however, are far from being so. Large corporations, such as those I have in mind, when planning any important innovation in practice or addition to plant, commonly commission their heads of departments to visit mines and works, often at very distant points, to seek information and to learn from the experience of others whose operations have been along the lines concerning which knowledge is desired. This entails a considerable expense, but its justification has been more than proved by the results obtained. In the same way, it would repay these corporations a thousandfold to take the trouble to interest their technical employees and foremen in the Institute and its work; and even if it were necessary for the corporations to pay the fees of certain members of their staffs and defray the expenses of some of their technical employees to enable them to attend the Annual Meetings, it would be greatly to their profit to do so. Again why should not the large mining corporations present a life-membership in the Institute as an award of merit to those of their employees who proved deserving? Such an action would demonstrate the company's appreciation of the usefulness of the Institute, while it would also be an appropriate means by which recognition of faithful and efficient service by an employee could be shown. Of corporations to which these suggestions might well be submitted for consideration, I could name readily at least three in Nova Scotia, three in Quebec, nine in Ontario, three in Alberta, and three in British Columbia. The addition of desirable men to our membership roll would necessarily strengthen our organization, and consequently increase our ability to assist towards the more efficient organization of our mining industries.

Since Canada became a nation the lack of co-ordination

has been the outstanding defect in our administrative methods. Its drawbacks, however, are more in evidence at the present time when the need for concerted action has become apparent and insistent. The more one studies the situation the more appallingly apparent becomes our unpreparedness. We have, for example, enormous masses of information, that would be of incalculable value to industry, buried in different Governmental Departments (both Federal and Provincial), in our Universities, and in corporation and private offices, but up to the present we have had no one possessing the necessary power or initiative to make it available so that it might be turned to practical account in the national interests. Our crying need is for co-ordination of effort and co-operation between the various agencies and organizations of the Dominion in the important work of striving to place the country in a more self-respecting position than it has occupied heretofore of being largely a hewer of wood and drawer of water for the United States. The great need is for a centralized direction. In the Mother Country a War Council of five members is supreme. Some such strong central organization is what we require in Canada to take command and to organize our industries. As staunch upholders of the principle of democratic freedom, we, as Canadians will take second place to the citizens of no other nation on earth; nevertheless, under the present conditions we would be ready even to acknowledge and follow a Dictator if such a one could prove his ability to re-organize effectually our national and industrial affairs and methods. At the same time we already have in the different Governmental Departments excellently qualified men who should be recognized as leaders in their respective fields. They should be given that opportunity; and if they failed to lead be replaced by others who would succeed. With a strong central Industrial Council armed with the necessary authority, the different Governmental departments could first be organized; and then societies, such as our own, and corporations and individuals ready to assist, could be directed what to do, so as to ensure concentration, conservation and real efficiency of effort towards the attainment of the desired end. About a year ago a Committee of the Institute submitted a report on the status of the Canadian mining industries to the Rt. Hon. the Minister of

Trade and Commerce. Embodied in this report were certain recommendations of a general nature, indirectly relating to, but nevertheless, importantly bearing on the subject of the report which dealt essentially with the possibilities as regards the utilization of our mineral resources to greater national advantage. One of these recommendations urged the imperative necessity of the appointment of a Commission to organize departments of the Civil Service for the purpose of correlating information respecting resources, trade and industry. Such a Commission has since been appointed, and we now look to it to accomplish successfully the task with which it has been entrusted. That task is as colossal as the need for its achievement is imperative.

SODIUM SULPHIDE PRECIPITATION OF SILVER AT THE NIPISSING MINE.

By R. B. Watson, Cobalt, Ont.

Annual Meeting, 1917

The effect of the war in changing existing conditions especially those relating to the supply and prices of materials, has been felt in many industries. This has not been wholly harmful for it has stimulated research for substitutes, some of which will be found better than those formerly in use and will remain in use after conditions have becomeagain normal. This is exemplified in the precipitation practice at the Nipissing low grade mill. E.M. Hamilton has described[1] aluminum dust precipitation at the Nipissing, and has shown that this method is not only cheaper than zinc dust precipitation, on account of the regeneration of cyanide, but that the latter method could not be used in this case on account of the deleterious effect of zinc and arsenic contained in the mill solutions. Before the war, aluminum dust cost 33.82c. and caustic soda 2.11c. per pound, laid down at the mine. At the expiration of our contract for aluminum dust in May 1916, the lowest price at which dust could be bought was 90c. per lb. and caustic soda had risen to 5.77 per lb. At these prices, the increased cost for these two chemicals would have amounted to about $33,000 per year. This was excessive and necessitated the finding of a substitute immediately. The utilization of sodium sulphide as a precipitant appeared to present the most promising solution of the difficulty; and the experiments carried out by J. J. Denny, in charge of the Company's research department, were so satisfactory, that in June, 1916, the process was adopted for the precipitation of all the mill solutions.

Hamilton suggested two objections to the use of sodium sulphide for this purpose: one, that complete precipitation could not be obtained without using an excess of the reagent; and the other, the difficulty of producing silver sulphide instead of metallic silver. As regards the first point, it was found in practice that the solution could be precipitated to .2 oz. silver without showing an excess of sodium sulphide in the barren solution. The

[1]Engineering and Mining Journal, May 10th, 1913.

second objection was met by converting the silver sulphide to bullion by using the same principle employed in the desulphurization of the raw ore before it is sent to cyanide treatment. This principle is that the various sulphide combinations of silver are decomposed by contact with metallic aluminum in a caustic soda solution: its successful application to the treatment of Nipissing ores was worked out by Denny and has been described by him.[2] This is the important feature of the process, since it gives fine bullion as a final product at a small cost. In the former days of the sodium hyposulphite process, before silver ores were treated by cyanide, the treatment and marketing of the 'sulphides' produced was the essential obstacle. Various methods were devised for the treatment of this product at the plant, but they were costly and unsatisfactory: the best of these, however, was the Dewey-Walter process in use at the Marsac mill, Park City, Utah.

The reactions involved are as follows:

1.—$2\ Na\ Ag\ Cy_2 + Na_2\ S = Ag_2\ S + 4\ Na\ Cy.$
2.—$2\ Al + 5\ Na\ OH + 3\ Ag_2S = Na_2\ Al_2\ O_4 + 6\ Ag + 3\ NaH\ S + H_2O.$
3.—$2\ Al + 8\ Na\ O\ H + 3\ Ag_2S = Na_2Al_2\ O_4 + 3\ Na_2S + 6\ Ag + 4\ H_2O.$

The commercial sodium sulphide used is supposed to be Na_2S plus 9 H_2O; this would be equivalent to 32.5% Na_2S. A concentrated form of sodium sulphide can be had, which promises to work out cheaper than the ordinary brand.

By equation (1) it would require .076 lb. avd. of commercial sodium sulphide to precipitate one troy ounce of silver. In practice the amount required is .12 lb.

In the desulphurizing operation as shown in equation (2) one troy ounce of silver requires .021 lb. of NaOH. This would be .028 lb. of commercial caustic soda carrying 76% NaOH.

If an excess of caustic is present in the operation, the reaction shown in (3) is apt to take place, in which case .034 lb. of NaOH, is used for each ounce of silver desulphurized. To prevent this taking place, the amount of silver present in the charge to be desulphurized is estimated as closely as possible and .03 lb. of commercial caustic soda is added for each ounce of silver contained.

[2]Mining and Scientific Press, September 27th, 1913.

The theoretical amount of aluminum necessary to desulphurize one ounce of silver is .0057 lb. avd.; the amount actually used in practice is .006 lb.

Apparatus Used.

The sodium sulphide is dissolved in an iron barrel, 2′6″ in diam. by 4′ deep, into which is flowing constantly a small stream of barren solution from the filter press. The barrel is divided by a vertical partition reaching nearly to the bottom: there is an excess of sodium sulphide in the barrel and the overflow pipe located on the other side of the partition from the inflow carries the precipitant to a tank, 5′ in diam. by 6′ deep, where it is agitated with the pregnant solution which is then pumped to the Merrill filter press. The amount of sodium sulphide added is regulated by a boy, who constantly tests the inflowing barren solution with pregnant solution and with sodium sulphide to determine whether there is an excess of sodium sulphide or unprecipitated silver present in the discharge from the press. The amount of barren solution coming into the barrel, and consequently the overflow of sodium sulphide, is regulated by a valve on the intake pipe in front of the operator. This operation could be carried out more precisely, perhaps, by precipitating a tankful at a time; but in our case, the tank capacity was not available The solution from the filter press is aerated thoroughly in the barren solution tank by air lifts and a large circulating pump, the object being to oxidize any sodium sulphide, should there happen to be a slight excess in the precipitated solution. The shift man always tests the solution for soluble sulphide before running it to the treatment tanks, and no bad effects have been noticed from this cause. Aeration is used in preference to lead salts as lead has a bad effect on extraction. A pressful of sulphide precipitate contains 25,000 oz. of silver. This is thrown into a tank 7′ in diam. by 5′ deep and the proper amount of caustic soda is added, figuring .03 lb. for each ounce of silver; the dilution is 4 parts of solution to 1 of sulphide; the strength of the solution in NaOH is about 8%. The pulp in the tank is kept mixed by the usual mechanical stirring device. Over this tank is mounted a small tube mill, 15″ in dia. by 6′ long, which makes 10 revolutions per minute. The charge

of aluminum ingots weighs 300 lb. and fills the tube mill half full. Below the tank is a centrifugal pump which circulates the pulp from the tank and through the tube mill for 10 or 12 hours. When the desulphurization is about complete the precipitate rubbed on a piece of paper by a spatula gives a silvery colour and a metallic lustre. It is then pumped to another press where it is washed for two hours; the resulting product is mixed with a small amount of borax, charged without drying into a reverberatory refining furnace, and melted down to bullion 996 fine. The slag is put through a small blast furnace.

The grade of the precipitate depends largely on how well the pregnant solution has been clarified before precipitation, but under ordinary conditions it assays about 23,500 oz. before and 26,500 oz. after desulphurization. The final product carries from ½% to 1% sulphur. Gold is not precipitated in an alkaline cyanide solution, and copper does not come down if there is sufficient free cyanide in the solution.

Analysis of Precipitate Lot No. 79.

	Before Desulphurizing per cent.	After Desulphurizing per cent.
Ag	81.21	91.72
S	12.09	.58
Soluble in water	2.02	.52
Insoluble in acid	2.63	3.03
FeO	.46	.57
CaO	.44	.49
Al_2O_3	1.32	1.55
MgO	.14	.16
Hg	.10	.11
	100.41	98.73

The precipitate contained no Au, Pb, Cu, Bi, Co, Ni, Sb, As or Zn.

Cost Compared with Aluminum Precipitation.

Considering the present war price of aluminum dust, the continued use of this reagent for precipitation is out of the question. It would appear, however, that even when prices of all chemicals used return to normal, sodium sulphide precipitation will be cheaper than the method formerly employed.

In the following comparison, pre-war prices are given: the quantity of aluminum dust used per ounce silver is the average for 1915.

	cents per oz. silver	
Aluminum Dust Method:		
.0262 lb. Aluminum dust @ 33.82c	.8861	
.05 " Caustic Soda @ 2.11c	.1055	
Labour, power, and work shops	.1710	
		1.1626
Sodium Sulphide Method:		
.12 lb. Commercial Sodium Suplphide @ 1.89c	.2268	
.006 " Aluminum Ingots @ 19.25c	.1155	
.03 " Caustic Soda @ 2.11c	.0633	
Labour, power and workshops	.3020	.7076
Difference in favour of Na_2S method		.4550c
Saving per year — 2,000,000 ounces @ .455c		$9,100.

It has been shown in Hamilton's article that a large part of the saving in the use of aluminum over zinc dust is in the regeneration of cyanide. By comparing equation (1) with Hamilton's equation (4) it will be seen that the regeneration is the same when sodium sulphide is used.

(4) $2\ NaAgCy_2 + 4\ NaOH + 2\ Al = 4\ NaCy + 2\ Ag + Na_2Al_2O_4 + 4\ H.$

The process as above described is limited in its application to silver ores, as gold sulphide is not precipitated. It is possible, however, that the above procedure could be used to precipitate the silver, followed by the precipitation of the gold in the solution by aluminum.

MILLING PRACTICE AT THE BUFFALO MINES, COBALT, ONT.

By Robert E. Dye.

Annual Meeting, Montreal, March, 1917.

The first serious attempt to apply flotation to the treatment of Cobalt ores was made in the fall of 1915, when the Buffalo Mines, Limited, under the management of Mr. Thos. R. Jones, installed a 50-ton Callow-flotation unit with which exhaustive test runs were made on mine rock and various mill products. These tests led to the construction of a 600-ton plant to treat the current sand tailing from tables as well as the sand tailing that had accumulated since milling operations started. The plant is now in successful operation.

It may be remarked here that milling practice in the Cobalt district varies greatly both as to methods of grinding and of subsequent treatment. Some mills adhere closely to a system of gravity concentration, in which the slime tailing is commonly discarded and the sand tailing stacked for further treatment. Others employ the all sliming and cyanide process. The practice at the Buffalo mine, up to the time of the introduction of flotation, was to crush the ore in stages to pass a 12-mesh screen and concentrate the coarse grades in jigs, and the sand by tabling. The slime formed in the mining operations as well as that formed in crushing the rock to pass 12-mesh was delivered to slime tables and the tailing from these further treated by cyanide; the cyanide tailing was then run to waste and the sand tailing from the concentration tables was stacked.

A large tonnage of sand tailing has thus accumulated and is available for retreatment. This product, much of which carries 5 to 6 oz. of silver per ton, but which in the early days of milling carried as high as 20 oz. per ton, constitutes an asset of considerable value. Further crushing followed by gravity concentration shows but little additional recovery of silver, hence it was most desirable to find a cheap method for the recovery of the silver in this low grade material. In 1914, tests on a working scale, demonstrated that this material could be treated profitably by cyanidatation.

In 1915 experimental work on flotation was commenced with a standard two-compartment cell, used as a rougher, and a half-size single compartment cell, used as a cleaner, both of the Callow type. The grinding was accomplished in an 8-ft. Hardinge mill. The experimental plant had a capacity of approximately 50 tons per day. Test runs on various mill products and various grades of ore were made daily for several months.

From the results of former tests on cyanide treatment and these tests on treatment by flotation, the probable performance and the cost of operation of plants of the two types were estimated as follows:—

CYANIDE PLANT, 500 TONS CAPACITY, OPERATION ONLY.

(Based on head assay of 6 oz. silver per ton.)

Handling sand	$0.040 per ton ore
Tube milling and classifying	0.500 " " "
Agitation, thickening, etc	0.160 " " "
Heat and light	0.091 " " "
Chemicals	0.487 " " "
Attendance and overhead expenses	0.140 " " "
Refining	0.030 " " "
Miscellaneous	0.172 " " "
Tailing loss 0.75 oz. at 56c	0.420 " " "
TOTAL COST	$2.040 per ton ore

FLOTATION PLANT, 600 TONS CAPACITY, OPERATION ONLY.

(Based on head assay of 6 oz. silver per ton.)

Handling sand	$0.040 per ton ore
Tube milling and classifying	0.400 " " "
Heat and light	0.030 " " "
Attendance and overhead expense	0.140 " " "
Cell treatment	0.080 " " "
Drying and loading concentrate	0.050 " " "
Smelter, including freight and losses	0.833 " " "
Tailing loss $\frac{3}{4}$ oz. at 56c	0.420 " " "
TOTAL COST	$1.993 per ton ore

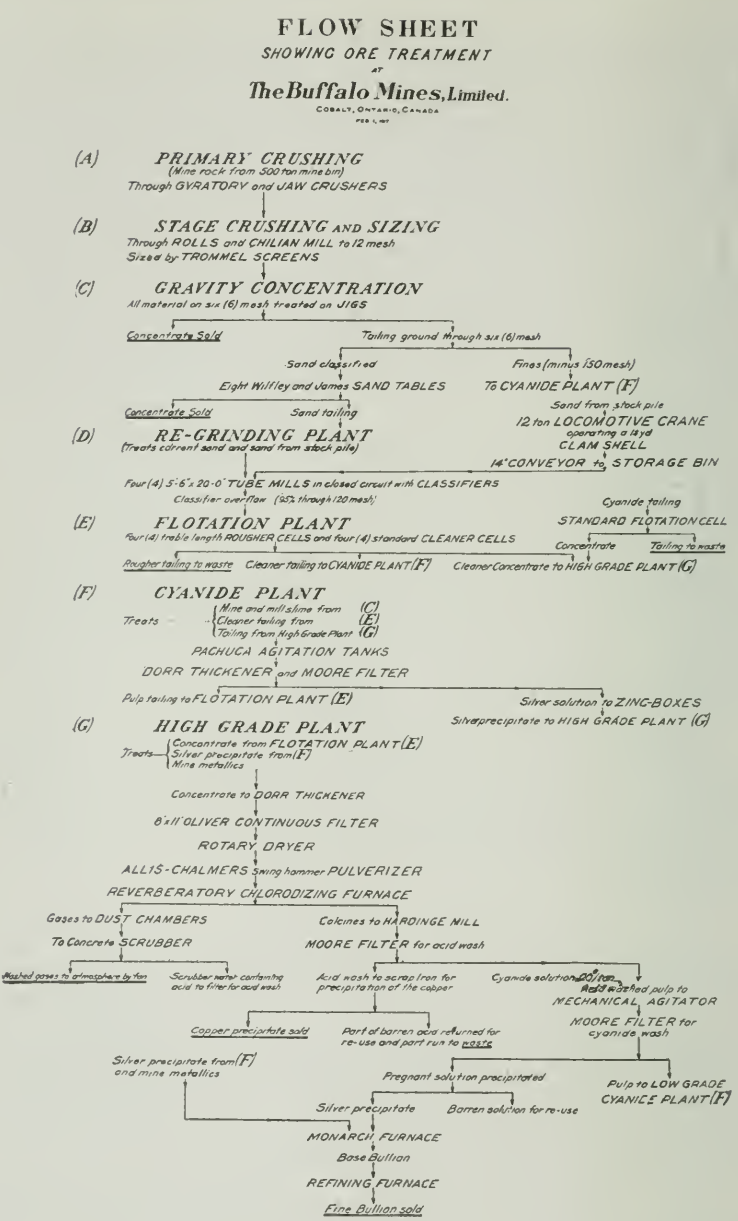
The above estimates cover operation costs only, and do not include the high rate of depreciation which should be written off against a milling plant put to such use. It may be remarked also that the costs of treatment by cyanide, as tabulated, were estimated when ante-bellum conditions prevailed, since when the cost of chemicals and supplies has advanced considerably.

In deciding to install a flotation plant, two considerations seemed to be of outstanding importance. One was that the installation of a flotation plant would greatly reduce the amortization charges, in that such a plant could be built much more cheaply. It was found possible to install the entire battery of cells, having 600 tons capacity, in the space formerly occupied by slime tables treating 50 tons per day. Of the four tube mills required, two were put into the old mill building, making it necessary to provide new building room for but two tube mills. The other factor was that of the $1.993 charged against flotation, 88.3c was for marketing concentrate. The few small tests made on the concentrate at that time indicated that there was an opportunity of greatly reducing this charge by treating the concentrate locally. With this information at hand and having in mind certain difficulties to be overcome, the Company erected the 600-ton flotation plant.

A general idea of the milling methods employed here is obtainable from the accompanying flow-sheet of the milling plant. It will be observed that the plants are in a closed circuit, and no final tailing is discharged except from the flotation plant. The important steps in the treatment are, in brief, as shown on following page.

(A) *Primary Crushing.*—In the primary crushing plant the run of mine rock is delivered from the 500-ton capacity mine rock storage bin to a No. 5 McCully gyratory crusher, the discharge from this crusher passing over a 30″ picking-belt to a trommel having $1\frac{1}{2}$″ round holes. The oversize from the trommel passes to a 20″ x 6″ jaw crusher of the Blake type. The discharge from this crusher joins the undersize from the trommel and is conveyed to the mill storage bin.

(B) *Stage Crushing and Sizing.*—The ore from the mill storage bin is fed to 30″ power and mining rolls set at $\frac{3}{4}$″. The


FLOW SHEET
SHOWING ORE TREATMENT
AT
The Buffalo Mines, Limited.
COBALT, ONTARIO, CANADA
(A) PRIMARY CRUSHING
(Mine rock from 500 ton mine bin)
Through GYRATORY and JAW CRUSHERS
(B) STAGE CRUSHING AND SIZING
Through ROLLS and CHILIAN MILL to 12 mesh
Sized by TROMMEL SCREENS
(C) GRAVITY CONCENTRATION
All material on six (6) mesh treated on JIGS
Concentrate Sold
Tailing ground through six (6) mesh
Sand classified
Fines (minus 150 mesh)
Eight Wilfley and James SAND TABLES
To CYANIDE PLANT (F)
Concentrate Sold
Sand tailing
Sand from stock pile
12 ton LOCOMOTIVE CRANE
operating a 1½ yd
CLAM SHELL
(D) RE-GRINDING PLANT
(Treats coarse sand and sand from stock pile)
14" CONVEYOR to STORAGE BIN
Four (4) 5'-6" x 20'-0" TUBE MILLS in closed circuit with CLASSIFIERS
Classifier overflow (65% through 120 mesh)
Cyanide tailing
(E) FLOTATION PLANT
STANDARD FLOTATION CELL
Four (4) treble length ROUGHER CELLS and four (4) standard CLEANER CELLS
Concentrate
Tailing to waste
Rougher tailing to waste
Cleaner tailing to CYANIDE PLANT (F)
Cleaner Concentrate to HIGH GRADE PLANT (G)
(F) CYANIDE PLANT
Treats
Mine and mill slime from (C)
Cleaner tailing from (E)
Tailing from High Grade Plant (G)
PACHUCA AGITATION TANKS
DORR THICKENER and MOORE FILTER
Pulp tailing to FLOTATION PLANT (E)
Silver solution to ZINC-BOXES
Silver precipitate to HIGH GRADE PLANT (G)
(G) HIGH GRADE PLANT
Treats
Concentrate from FLOTATION PLANT (E)
Silver precipitate from (F)
Mine metallics
Concentrate to DORR THICKENER
8'x11' OLIVER CONTINUOUS FILTER
ROTARY DRYER
ALLIS-CHALMERS Swing hammer PULVERIZER
REVERBERATORY CHLORODIZING FURNACE
Gases to DUST CHAMBERS
Calcines to HARDINGE MILL
To Concrete SCRUBBER
MOORE FILTER for acid wash
Washed gases to atmosphere by fan
Scrubber water containing acid to filter for acid wash
Acid wash to scrap iron for precipitation of the copper
Cyanide solution
Acid washed pulp to
MECHANICAL AGITATOR
MOORE FILTER for cyanide wash
Copper precipitate sold
Part of barren acid returned for re-use and part run to waste
Silver precipitate from (F) and mine metallics
Pregnant solution precipitated
Pulp to LOW GRADE CYANICE PLANT (F)
Silver precipitate
Barren solution for re-use
MONARCH FURNACE
Base Bullion
REFINING FURNACE
Fine Bullion sold

discharge from this set of rolls is sized to four products by means of trommels. That portion on screen of $\frac{3}{8}''$ mesh goes to bull jigs. The portion through $\frac{3}{8}''$ and on 6-mesh goes to fine jigs. The product through 6-mesh and on 12-mesh passes to the Chilian mill for further grinding. The product through 12-mesh goes direct to a hydraulic classifier for table feed.

The tailing from the bull jigs is sized in a trommel with $\frac{1}{2}''$ openings, the oversize returned to mill bin and the undersize passing to a set of Allis-Chalmers fine grinding rolls. The discharge from this set of rolls joins the discharge from the Power and Mining rolls and again passes through the sizing circuit.

The minus $\frac{3}{8}''$ plus 6-mesh tailing from the fine jigs, passes through a dewatering screen and is divided, one portion being sent to fine rolls and the remaining portion to the Chilian mill. The Chilian mill is a 6-ft. Evans-Waddell and is fitted with a 12-mesh screen. The discharge from this mill joins the discharge from the rolls and is returned to the sizing trommels. It may be stated that this arrangement is made for mechanical reasons only, since the Chilian mill discharge is of proper size to go direct to the hydraulic classifier.

Stage crushing as it is now conducted is probably not so essential as it was formerly when the plant depended largely upon gravity concentration for the recovery of the silver contained in the ore. At that time it was important to avoid, as much as possible, the sliming of the valuable minerals. At the present time a ball mill and classifier would probably displace to advantage the rolls, Chilian mill, and trommels, since with the present mode of treatment the production of fines is not considered a serious disadvantage.

(C) *Gravity Concentration.*—The concentrate from the jigs is taken from the jig beds at intervals and is shipped direct to the smelter, since no provision is now made for local treatment. The hutch product is sent to the Chilian mill. The tailing from the jigs is crushed through 12-mesh as shown above and is then conducted to hydraulic classifier furnishing the feed to eight sand tables of Wilfley and James type. The minus 12-mesh sand is classified in a hydraulic classifier, the products from the spigots forming table feed and the overflow going to the cyanide

plant. This overflow product is made up of slime formed in mining and the slime formed in reducing the ore to 12-mesh. It is found that the mine slime is not readily treated by the flotation process. By sending the mine slime, together with the mill slime, to the cyanide plant, a greater ultimate recovery is obtained, and the resulting silver is secured in the form of bullion.

The tables make two products, concentrate and tailing, the usual middling product being included in the tailing. The table tailing is sent to the re-grinding plant to be ground preparatory to treatment by flotation. The concentrate from the tables is sold to a smelter at present. The grade of the table concentrate has been materially raised by using, on the tables, the water overflowing the Dorr thickener to which the cleaner tailing is pumped. This water contains a small amount of oil, which tends to carry a large part of the barren pyrite and other fine sulphide minerals into the table tailing. These sulphides are later recovered in the flotation concentrate. It is a decided advantage to collect the pyrite and other sulphides, with the flotation concentrate, since sulphur is essential for carrying on the chloridizing roast, later described.

(D) *Re-Grinding Plant.*—The re-grinding plant consists of four 5′ 6″ x 20′ 0″ Power and Mining tube mills, each of which is operated in a closed circuit with a standard duplex Dorr classifier. Three of the mills are each driven by a 100 hp., 2,200 volt, 60 cycle motor. The fourth mill is driven by a 125 hp., 550 volt, 60 cycle motor. The power required for each mill is approximately 85 hp. Each mill handles 125 tons of sand per 24 hours. About 1% of the tube mill feed stays on 10-mesh and about 6.5% passes 100-mesh. The discharge from the classifiers carries 5% on 120-mesh. One of these mills handles the current sand tailing from the concentration plant. The other three mills operate on sand from the stock pile.

The sand from the stock pile is handled by a 12-ton Brown-locomotive crane operating a $1\frac{1}{4}$-yd. clam shell bucket. The sand is delivered to a short 20″ conveyor belt by means of a loading hopper, and is conveyed to a sand storage bin by a system of 14″ conveyor belts. From the storage bin the sand is fed

continuously by means of a screw feeder to a 14″ conveyor belt delivering the feed to a sand splitter, which distributes an equal portion of sand to each of the mills.

(E) *Flotation Plant.*—All the flotation machines used are of the Callow pneumatic type. The flotation plant consists of four two compartment, treble length, rougher cells of a special design, and four standard cleaner cells. The air required, under a pressure of 5 lb. per sq. inch, is furnished by three high pressure rotary blowers of the Root type, each having a capacity of 2,300 cu. ft. of free air per minute. Each blower is driven by a 50 hp. motor, two blowers being sufficient to furnish air for the entire plant, the third always being in reserve as a spare.

The rougher cells used were developed to conform to the peculiar conditions prevailing in the Cobalt district. Results from the experimental plant showed that relatively long treatment in the roughers was necessary; that is, best results were obtained when the pulp was passed through two or even three standard roughing cells. The fact that this ore must be crushed to 100-mesh and finer to free the silver values made the usual steep bottom, as used in the standard cells, unnecessary. To avoid passing the pulp through three standard cells, a long cell was constructed three times the standard length, with a slope per foot of one-third that given the bottom of a standard cell. This made a more compact installation, and since the headroom available was limited, avoided the pumping of the pulp from each rougher cell to the next of the series. The nominal capacity of each compartment of the treble length cells is 75 tons. It is found that each compartment will easily handle 100 tons per 24 hours; hence more than three rougher cells are never in operation at one time.

Owing to the fact that marketing charges are very high. The production of a high-grade concentrate from the Cobalt ores is more essential than at most of the copper camps where flotation has been so successfully employed. The high freight rates are due to the long haul, and the high smelter charges are ascribable largely to the extreme fineness of the product shipped. With a standard cleaner cell a high concentrate invariably results in the production of a tailing of high value. This diffi-

culty was overcome to a large extent, by a device perfected by C. J. B. Armstrong while in the employ of the Buffalo mine. Essentially, the idea is to restrict the flow of the froth from the cell, drawing off the concentrate at only one or two points. Thus, instead of drawing off a thin film of froth along the full length of the cell, as is ordinarily done, the froth is drawn off in deep narrow streams through notches. This is accomplished by building up both sides of the cell along its entire length, and cutting notches at the points of overflow. This device enables the operator to draw off a small steady stream of high-grade concentrate with more ease and certainty than is attainable with a standard equipment. The device is used to great advantage on the cleaner cells.

In the operation of the plant the rougher cells produce two products, the rougher concentrate and the tailing. The tailing is sent directly to waste and the rougher concentrate is collected and split into four parts, each part being sent to one compartment of a standard cleaner cell. The cleaner cell makes two products, the concentrate and a tailing or 'flotation middling'. The concentrate is pumped direct to the 'High Grade' plant for treatment. The cleaner tailing or 'flotation middling' is sent to the 'Low Grade' cyanide plant. The so-called 'flotation middling' is probably not a true middling, but is made up largely of a mass of imperfectly oiled particles.

The frothing agents used to accomplish separation by flotation are added to the feed going to the tube mills. In this way the mills act as emulsifiers as well as grinders. Little or no tar, with a large proportion of creosote and a small amount of pine oil, make up a part of the frothing mixture. It is found that such a mixture produces a froth from which a high concentrate can be made in the 'cleaner' cells and that the tailing (flotation middling) from these can readily be treated by cyanide. This is not true if a greater proportion of tar is used, since the tar produces a lower grade concentrate and interferes with the cyanide treatment to a much greater extent than does the creosote. It is found that, by treating with cyanide the small amount of 'flotation middling' formed, a greater ultimate extraction is obtained and a greater profit results. The tailing from the cleaners may, however, be treated to fair advantage by return-

ing it to the flotation circuit. In this connection it may be stated that owing to the injurious effect of the turpentines and light oils on the workmen, it is not advisable to use ingredients in the frothing mixture which have a distillation point of less than 190°C.

The tailing coming from the cyanide plant, which is also treated by flotation, is not mixed with the general flotation feed, but is treated in two separate cells. The concentrate formed is run with the concentrate from the cleaner cells, without cleaning, and is pumped to the 'High Grade' plant. The tailing resulting is sent directly to waste.

(F) *Low Grade Cyanide Plant.*—The 'Low Grade' cyanide plant treats the slime formed in mining and in crushing the rock to 12-mesh, the tailing from the cleaner cells, and the tailing from the 'High Grade' plant resulting from the treatment of the flotation concentrate. While all the products treated show a profit over treatment by flotation, the cyanide plant probably has its greatest use in serving as a safety for the 'High Grade' plant; that is, the dissolved values in the tailing coming from the 'High Grade' plant are largely due to the fact that facilities for washing the pulp there are limited. These dissolved values are saved and a slight additional extraction is effected by re-agitation and filtration.

The slime from the concentrator and the 'flotation middling' are pumped to a 30′ x 14′ Dorr thickener, the overflow water being used again for flotation and as wash water on the sand tables and the underflow elevated to a Pachuca tank for storage. This pulp is dewatered on a Moore filter, cyanide solution added, and is then pumped to Pachuca tanks for agitation. The tailing from the 'High Grade' plant is sent to the cyanide plant in a cyanide solution, pumped directly to the Pachuca agitators and treated with the raw pulp. The cyanide solution in the agitators is made up to 8.5 lb. potassium cyanide per ton at the start and the protective alkali is kept at 0.5 lb. lime per ton. After 60 hours agitation the entire contents of the agitator is delivered to a 30′ x 10′ Dorr thickener and barren solution added. The underflow from the thickener is sent to a Dorr agitator, which serves as a storage tank for pulp going to the Moore filter. The chemical consumption of cyanide is 5.5 lb. and the mechanical

loss is 9/10 lb. per ton of pulp. The average consumption of lime in the agitators is about 2.7 lb. per ton of pulp treated. This high consumption is due largely to the oil in the product coming from the cleaner cells. Approximately, the lime consumed by the different products is: for the mine and mill slime, 0.2 lb. per ton, for the cleaner tailing from flotation, 4.0 lb. per ton, and for the tailing from the 'High Grade' plant, 1.0 lb. per ton of pulp. The clear pregnant cyanide solution is decanted from the thickener and passed through zinc-boxes for precipitation. The pulp is filtered and washed on a Moore filter in the usual way, the solution being sent to the zinc-boxes. The discharge pulp from the filter is emulsified with enough water to bring it to a consistency of about six parts of water to one part of dry slime by weight, and is sent to the flotation plant.

(G) *High Grade Plant.*—As already mentioned, it is very desirable to treat the flotation concentrate by some cheap local method and thus, in a measure, avoid the high marketing charges demanded for smelting flotation concentrate. Looking to the accomplishment of this, much research work has been done, while at the same time the tests have been in general on a commercial scale. Since the silver in the flotation concentrate does not yield to direct cyanide treatment, it is necessary, in order to recover the silver by cyanide, to render it amenable by some previous treatment. Of the various methods of treatment tried, a chloridizing roast seems to be the most promising. Experimental work in this connection was first carried out in a 2′ x 4′ reverberatory roasting furnace. The results of a number of these test runs are tabulated below:—

EXPERIMENTAL RESULTS ON THE CHLORIDIZATION OF FLOTATION CONCENTRATES.

Roast No.	Raw Assay	% NaCl	% Sulphur	Roasted Assay	Vol. Loss %	KCN Cons.	Pulp Tails	Extr'n on Rstd.	Actual Extraction %
1	235.2	15%	4	196.6	8	18	13.0	93%	85
4	235.2	15%	2	185.0	12	16	9.0	95	83
12	300.0	15	2	260.0	5	15	12.0	90	90
16	204.0	9	2	177.0	6	10	10.0	94	89 acid wash
A	270.0	5	3	235.0	9	10	9.0	96	87 acid wash

In the above experiments, charges of 100 lb. each were used, the charge being spread evenly over the hearth after adding the salt and sulphur as indicated. The charge was rabbled at short intervals during the roast. The time required to complete the roast was about eight hours, though the charge was allowed to cool over night in the furnace. In these tests the temperature was rapidily raised to 400°C and held there for five or six hours, after which the temperature was raised to 600°C to complete the chloridizing roast. The tabulated results indicate an extraction of less than 90% on the raw ore, although the extraction on the roasted product was 95%. It may be assumed that by the installation of proper means for preventing dust losses, the dust losses in the test runs could be materially reduced. The encouraging nature of the preliminary results led to the construction of a larger experimental furnace of the same type, the intention being to conduct further experiments on a larger scale. This furnace is still in operation, and the process used at present is as follows:—

The concentrate from the flotation plant is pumped to the 'High Grade' plant by a 5″ x 5″ Aldrich triplex pump, after the addition of approximately 30 lb. of lime per ton of concentrate. The pulp discharges to a 20′ x 12′ Dorr thickener, the overflow water being run to waste. The thickener feed carries one part of solids to twenty-five parts of water by weight. The underflow carries about 50% moisture. The Oliver filter, used for de-watering, is equipped with a flapper by means of which an extra amount of from 5% to 8% of the moisture in the cake is eliminated. A cake carrying 25% moisture is discharged from the filter. About 50% of the salt required for the roast is added to the filter cake as a brine wash which displaces the water in the cake, the aim being to add enough brine to permeate the cake and yet avoid any salt loss through drawing the brine solution through the cake and into the vacuum pump. This gives an excellent mixture of the salt with the pulp. The cake from the Oliver filter is sent to an oil-fired, direct-heat, counter-current, revolving dryer, which reduces the moisture content to about 10%. This dryer consists of a steel drum 30′ x 3′ 6″ lined with fire brick, and was made from the shell of a Burt revolving filter. The escaping gases are sent through a

cyclone dust collector and are discharged by means of an exhaust fan. The material from the dryer, after the addition of the remaining salt and sulphur required, is put through an Allis-Chalmers swing hammer pulverizer and sent to a storage bin from which it is drawn to the roasting furnace.

The roasting furnace is a double hearth, oil-fired, hand-rabbled, reverberatory-roaster, each hearth being 8′ x 22′. The fire box is at one end of the lower hearth, the flame passing over the lower hearth and then up and back over the upper hearth. The raw ore is charged at the front end of the upper hearth and is rabbled to the back end of the furnace where it falls to the lower hearth. It is then rabbled along the lower hearth to the fire box end, where it drops into a cooling pit provided for the roasted ore. The furnace is rabbled at short intervals, and every two hours a quantity of roasted ore is discharged into the pit and more raw ore started through the furnace. The furnace will handle about three tons of raw ore per 24 hours.

The gases from the furnace pass through a system of dust chambers to a water-sprayed gas scrubber, and are then exhausted by an acid-proof fan. The gases first pass to a dust chamber 6′ x 6′ x 7′ 6″ high, divided into two compartments. This chamber catches most of the heavy dust particles. The gases are then conducted to a dust chamber 6′ x 13′ x 6′ 6″ high divided into three compartments; thence the gases pass through a 30″ cooling pipe to a concrete flue conducting them into the bottom of a reinforced concrete scrubber, 6′ x 6′ x 16′ high inside. This scrubber is filled with a network of 6″ partition tile over which water flows. The scrubber water flows down the concrete gas flume to an acid-proof centrifugal pump made of Duriron, which keeps it in constant circulation. When the percentage of hydrochloric acid in the scrubber water has increased sufficiently, it is diverted to the Moore filter where it is used for the acid wash given the roasted ore. The escaping gases are very corrosive and rapidly destroy any iron with which they come in contact. The exhaust fan used is made entirely of wood except for the shaft, the housing and impeller being so constructed that no bolts or nails are exposed to the action of the gas handled. A wooden stack of similar construction is also provided for the escaping gases.

The calcines from the roaster are put through a 6′ wet-grinding, Hardinge pebble mill to break up any lumps present. A cake from the pulp, is then formed on a Moore filter, and is given an acid wash with a 0.3% hydrochloric acid solution. The solution resulting from this acid wash, which at times carries as high as 0.25% copper, is sent to scrap iron precipitation boxes for the recovery of the copper. A part of the barren solution resulting is continuously run to waste and a portion returned for re-use. The strength of the acid wash is kept up by the addition of hydrochloric acid and by using the acid water obtained from the scrubber. Following the acid wash the pulp is thoroughly washed with water to free it from acid.

Considerable difficulty is experienced in giving the pulp an acid wash on the Moore filter, since the copper content in the solution makes it very active in attacking all iron parts. Some difficulty has been encountered in securing a vacuum pump to withstand the action of the acid. A rotary pump is to take the place of the reciprocating pump now in use. Duriron (a high silicon iron) and pure copper, seem to be the only suitable materials which withstand the action of the acid solution. Duriron is quite brittle and cannot be put to every use, though where it can be employed it gives very satisfactory results.

A Moore filter basket was constructed using a copper manifold and a smooth bore vacuum hose for conducting the solution from the filter. The customary iron pipe for supporting the canvas of each leaf was replaced by a wooden top and frame of 1″ x 3″ oak. The solution is drawn from the bottom of the leaf inside the canvas in ordinary 1″ water hose carried over a $\frac{3}{4}$″ copper nipple, whence it passes through the wooden top-frame. The copper nipple is connected to the manifold by means of a short piece of rubber hose.

The washed pulp is dumped into a chute which conducts it to a mechanical agitator. Here cyanide solution of 15 lb. per ton strength is added, bringing the consistency of the pulp to about one to one by weight. The pulp is at once filtered on a Moore filter and given a cyanide wash in barren solution of the above strength. It is not necessary to allow any time for dissolving the silver, since the silver chloride present goes into solution almost instantly. After washing the pulp with four

parts of cyanide solution to one of pulp by weight, the cake is discharged and emulsified in a low-grade cyanide solution and sent to the 'Low Grade' cyanide plant for further treatment as previously described. The silver in the pregnant cyanide solution resulting from this treatment is precipitated in batch lots in a mechanically agitated tank, by means of sodium sulphide. The precipitate is collected in a plate and frame filter press. The metalizing of the precipitate can be satisfactorily effected by a wet desulphurization process[1], although if the desulphurization is not complete the refining of the resulting silver cannot be accomplished without considerable loss.

It may be of interest to remark that this silver sulphide precipitate when subjected to an oxidizing roast, conducted at a temperature of about 400°C, yields the metal in a mass of wire silver. Small samples of fused silver sulphide and of argentite subjected to a similar roast for a few hours became covered with numerous sprouts of wire silver, very similar in appearance to those occurring in nature. This phenomenon may be of interest to geologists and suggest a new theory concerning the deposition of native silver in nature.

The final refining of the silver is accomplished in an oil-fired reverberatory furnace having a capacity of 28,000 fine ounces of silver per charge. A charge containing little or no copper can be bought to a fineness of 996 or better in from fourteen to twenty-four hours, depending upon the size of the charge, the grade of the base bullion charged, and the quantity and nature of the impurities in the base bullion. The fine bullion produced except for copper, is free, practically, from impurity. It contains, however, about 80% of the total copper which enters the furnace with the charge.

The treatment at the 'High Grade' plant, viewed from the standpoint of extraction, and in the light of the mechanical improvements which can be made, is successful commercially.

A 7′ x 9′ Holt-Dern roasting furnace to replace the present roaster has been ordered and will be installed at an early date. This furnace, which is described in some detail in a paper submitted by Mr. E. B. Thornhill before this meeting, will greatly

[1]"Sodium Sulphide Precipitation of Silver at the Nipissing Mine," by R. B. Watson, Bull. C. M. I. No. 55, Nov. 1916. See also pp. 6-10 this volume.

lower the cost of the roast by cutting down the fuel required, and also by reducing the loss sustained through dusting and volatilization. It is anticipated, moreover, that the roasted product resulting from this furnace will be amenable to treatment by direct leaching in tanks, in which case the trouble and expense of the preliminary acid wash and filtration, will be eliminated. In view of the high cost of cyanide and the uncertainty of securing an adequate supply regularly the possibility of using an acid-salt leach for the removal of the copper and for the recovery of a part of the silver appears especially attractive. If this were done it would probably prove advisable to follow the acid-salt leach by a thorough water wash, and then re-grind the pulp and finish by cyanide treatment. These possibilities are suppositions based on uncertainties, and cannot definitely be decided for some time.

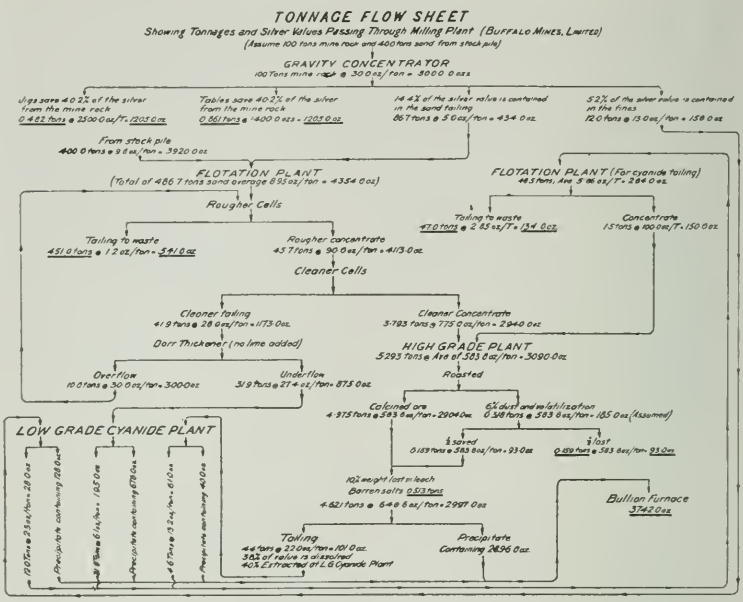
Assays of the 'Flotation Plant' products, for the week ending Feb. 10th, 1917 are tabulated below:—

	DAY				NIGHT			
	Flo. Feed	Rougher Tailing	Cleaner Tailing	Concentrate	Flo. Feed	Rougher Tailing	Cleaner Tailing	Concentrate
5th.....	9.4	1.3	22.0	462.0	7.6	0.8	18.0	499.0
6th.....	8.6	0.9	17.6	520.0	10.0	1.2	30.6	740.0
7th.....	8.4	1.2	14.0	688.0	7.8	1.1	24.0	1013.0
8th.....	10.0	1.2	16.6	760.0	9.0	1.1	59.0	775.0
9th.....	10.0	1.4	34.0	975.0	9.8	1.0	24.6	1085.0
10th.....	8.4	1.5	23.6	842.0	8.8	1.7	36.0	740.0
Average...	9.13	1.25	21.3	708.0	8.83	1.17	32.0	809.0

From the above, the averages for the week are:—

Feed assay..... 8.93 oz. per ton
Rougher tailing assay..... 1.20 " " "
Cleaner tailing..... 28.40 " " "
Cleaner concentrate..... 775.00 " " "

In obtaining the foregoing general averages for the week the night samples are given twice as much weight as the day samples, since they represent two shifts' run in each case, while the day samples represent but one shift's run.


TONNAGE FLOW SHEET
Showing Tonnages and Silver Values Passing Through Milling Plant (BUFFALO MINES, LIMITED)
(Assume 100 tons mine rock and 400 tons sand from stock pile)
GRAVITY CONCENTRATOR
100 Tons mine rock @ 30.0 oz/ton = 3000.0 ozs
Jigs save 40.2% of the silver from the mine rock
0.482 tons @ 2500 oz/T. 1205.0 oz
Tables save 40.2% of the silver from the mine rock
0.561 tons @ 1400.0 ozs. 1205.0 oz
14.4% of the silver value is contained in the sand tailing
86.7 tons @ 5.0 oz/ton = 434.0 oz
5.2% of the silver value is contained in the fines
12.0 tons @ 13.0 oz/ton = 156.0 oz
From stock pile
400.0 tons @ 9.8 oz/ton = 3920.0 oz
FLOTATION PLANT
(Total of 486.7 tons sand average 8.95 oz/ton = 4354.0 oz)
FLOTATION PLANT (For cyanide tailing)
44.5 tons, Ave. 5.86 oz/T. = 264.0 oz.
Roughter Cells
Tailing to waste
47.0 tons @ 2.85 oz/T. 134.0 oz.
Concentrate
1.5 tons @ 100.0 oz/T. 150.0 oz.
Tailing to waste
451.0 tons @ 1.2 oz/ton = 541.0 oz
Roughter concentrate
43.7 tons @ 90.0 oz/ton = 4113.0 oz
Cleaner Cells
Cleaner tailing
41.9 tons @ 28.0 oz/ton = 1173.0 oz.
Cleaner Concentrate
3.783 tons @ 775.0 oz/ton = 2940.0 oz.
Dorr Thickener (no lime added)
HIGH GRADE PLANT
5.293 tons @ Ave of 503.0 oz/ton = 3090.0 oz.
Over-flow
10.0 tons @ 30.0 oz/ton = 300.0 oz
Under-flow
31.9 tons @ 27.4 oz/ton = 875.0 oz
Roasted
Calcined ore
4.975 tons @ 593.0 oz/ton = 2904.0 oz.
6% dust and volatilization
0.348 tons @ 593.0 oz/ton = 185.0 oz. (Assumed)
LOW GRADE CYANIDE PLANT
1/2 saved
0.153 tons @ 585.0 oz/ton = 93.0 oz
1/2 lost
0.489 tons @ 585.0 oz/ton = 93.0 oz.
10% weight lost in leach
Barren salts 0.513 tons
4.621 tons @ 646.6 oz/ton = 2987.0 oz
Bullion Furnace
3742 oz.
Tailing
44 tons @ 2.2 oz/ton = 101.0 oz.
3.62 oz of value is dissolved
40% Extracted at L.G. Cyanide Plant
Precipitate
Containing 2896.0 oz.

Probably a better idea of what is accomplished in the plant will be gained by examining the accompanying flow-sheet in which the values of the grades in the various stages of the treatment, may be noted.

The value of the material treated at the time the data for the flow-sheet was collected is above the average. As a result the silver in the tailing is higher in value than would be the case if average ore was treated although the percentage extraction is about normal. Owing to the limited capacity of the 'High Grade' plant, it is at present necessary to make a small tonnage of high-grade concentrate. This results in the production of a 'cleaner' concentrate of relatively high value which yields a smaller proportion of its silver to raw cyanide treatment than does a middling of lower grade. The best results will probably

be obtained when it is possible to produce a 'rougher' concentrate of 60 to 70 oz. per ton, a 'cleaner' concentrate of 100 to 200 oz. per ton and a 'cleaner' tailing of 15 to 18 oz. per ton. This will be done when the capacity of the 'High Grade' plant shall have been increased. The flow-sheet shows that at the present time the total extraction from mine rock and from stock-pile sand, in the stated proportions, is 88.9%. The extraction on mine rock alone is 96.7% and the extraction on material from the stock pile is 82.8%.

PRESENT FLOTATION PRACTICE ON COBALT ORES.

By J. M. Callow and E. B. Thornhill.

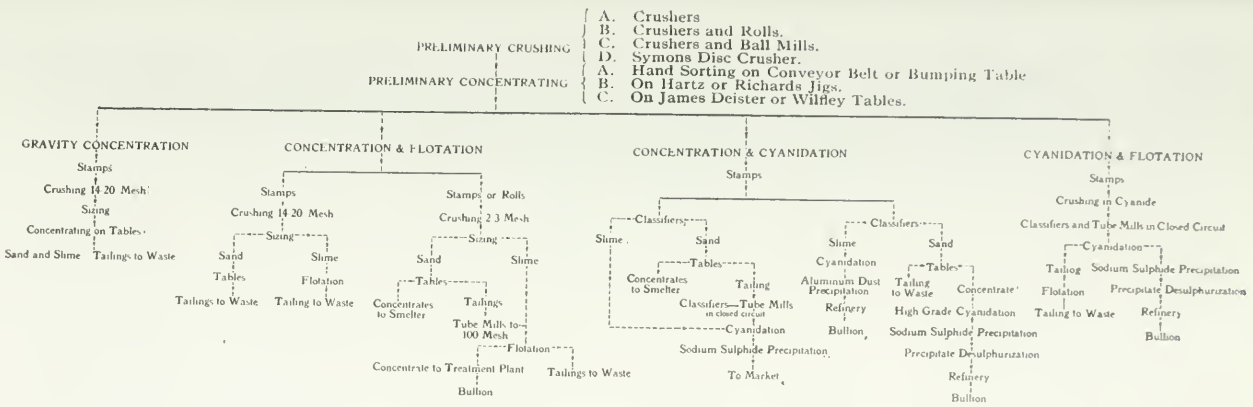
Annual Meeting, Montreal, March, 1917.

Prior to 1915 flotation as a means of saving the silver values in the Cobalt ores, not recoverable by gravity methods, had been experimented with to some extent on a small scale by several of the Cobalt mill operators. In October, 1915, however, the first experimental plant, operating on a tonnage scale, was built and operated by the Buffalo Mines Limited. This plant consisted of a two-compartment standard length Callow rougher cell, and one one-half size Callow cleaner cell. It was built primarily to determine the feasibility of concentrating the silver values in the stacked sand tailing from previous concentrating operations.

The results obtained with this experimental plant led the Buffalo Mines, Limited, to erect a Callow flotation plant of 600 tons daily capacity to treat the sand tailing, and to serve as an auxiliary in the treatment of its low-grade ores. This plant was put in operation in September, 1916. The McKinley-Darragh-Savage Mines also installed a Callow flotation plant of 200 tons daily capacity to treat the slime made in the mining and milling of its ores. This plant was put in operation in June, 1916.

The performance of these two initial plants has satisfactorily demonstrated on a commercial scale the adaptability of the flotation process to the low-grade ores and mill products of the Cobalt district. The present users of the Callow pneumatic system of flotation in the Cobalt district are the Buffalo Mines, MacKinley-Darragh-Savage Mines, the Nipissing Mines, the Coniagas Mines, the Dominion Reduction Co., the Northern Customs Concentrators, and the National Mines.

Flow-sheet No. 1, compiled by Fraser D. Reid, shows the milling practice at the different mills in the Cobalt district at



Mills	Equipment	Tn'ge
Hudson Bay	20 stamps	65
Trethewey	30 stamps	100
Temiskaming	30 stamps	100
Penn Canadian	Rolls-Pebble Mill	110
	Concentrates from mills shipped to Smelter	
Coniagas	60 stamps	180
McKinley Darragh	50 stamps	175
Northern Customs	80 stamps	200
National	Ball Mill	75

Mills	Equipment	Ton'ge
Beaver	Ball and Pebble Mills	130
	Concentrates from mills shipped to Smelter	
Buffalo (Mine Run of Ore and Tailings Pile)	Rolls and Chilean Mills	600
Dominion Reduction	40 stamps	200
Dominion Reduction trying out this process although equipped with Cyanide Plant		
O'Brien	30 stamps	170
Cobalt Reduction	120 stamps	275
Nipissing	40 stamps	240

FLOW SHEET No. 1.

(Comparative Flow Sheet of Cobalt Milling Practice, compiled by Fraser D. Reid).

the present time. This, when compared with the previous flow-sheet published in the Transactions of the Canadian Mining Institute for the year 1914, shows also the change in the metallurgy and equipment since that time.

The flotation process as adapted to the treatment of Cobalt silver ores has not replaced standard gravity concentrating machines to any extent as in copper milling, but its particular function is to save the more friable silver minerals and fine leaf silver that escapes recovery by gravity concentrating machinery.

Flow-sheet No. 2 shows the disposition of the flotation plant in its relation to the general scheme of treatment.

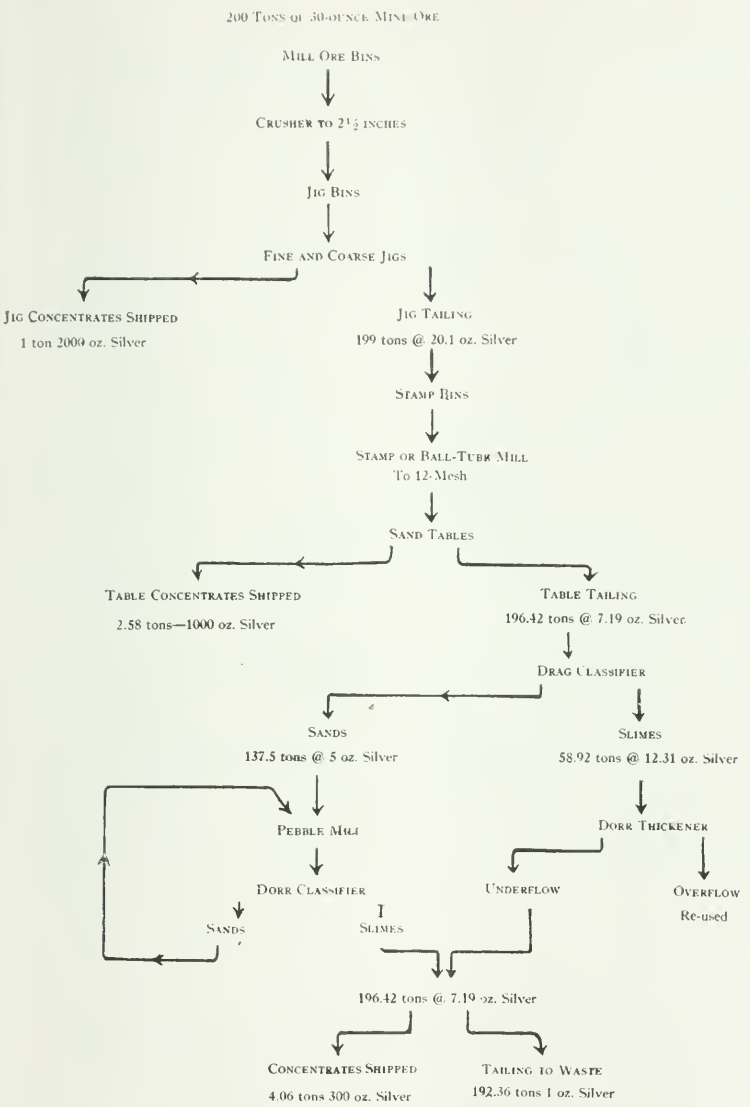
The treatment in the flotation plant proper is indicated by flow-sheet No. 3.

Comparatively fine grinding is required to liberate the values from the gangue. The economic limit of grinding for flotation appears to be reached when not more than 1% or 2% is retained on a 100-mesh screen.

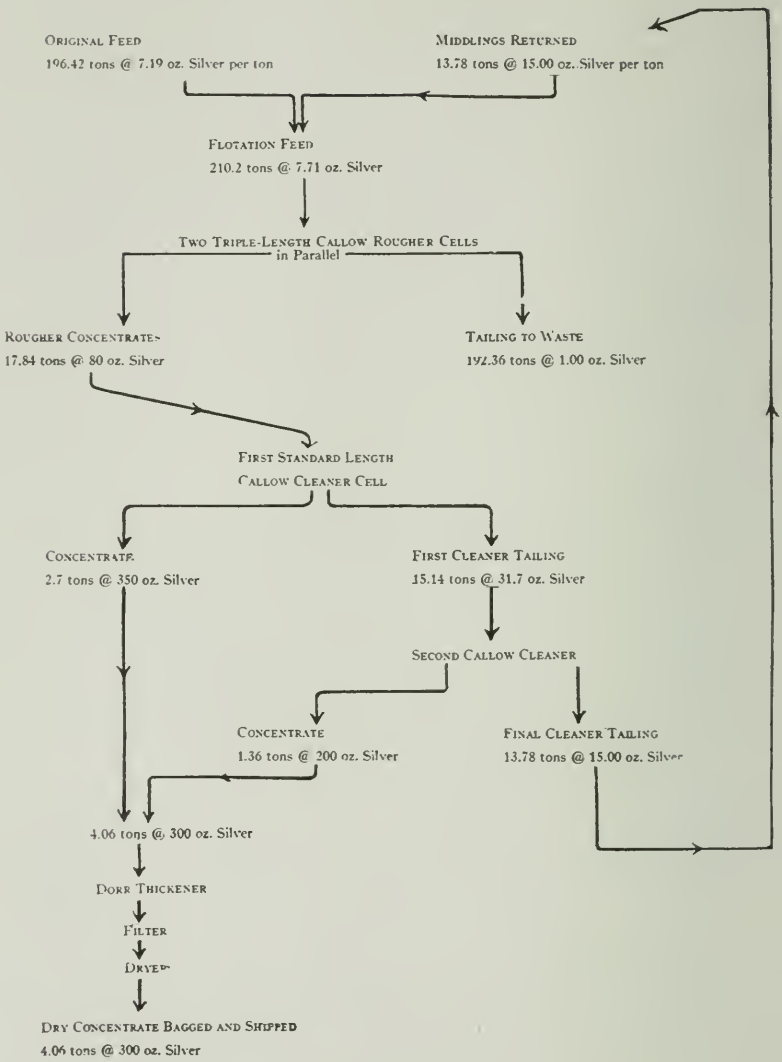
Practically all the plants use an oil mixture consisting of pine oil, coal tar creosote and coal tar. A mixture of 15% pine oil, 75% coal tar creosote and 10% coal tar is very efficient. A 10% pine oil and 90% high sulphur fuel oil mixture is being used with fairly good results to float cyanide plant tailing.

Recoveries by flotation vary over a rather wide range at the different plants, depending on the product going to flotation, the grade of concentrate required, and the experience of the operators. The last factor is probably the most important at present, as it requires time to produce an efficient operator.

On page 6 is the tabulated value of the heading, tailing, and concentrate, and the percentage recovery at the McKinley-Darragh-Savage Mines from mine and mill slime for the months of December, 1916, and January 1917. Previous results at this plant appear in Mr. J. M. Callow's paper presented at the New York meeting of the American Institute of Mining Engineers in February, 1917.



Flow Sheet No. 2.



Flow Sheet No. 3

Flotation Results at the McKinley-Darragh-Savage Mines Limited, Averages by Quarter Month Periods.

Month	Period	Heading	Tailing	Concentrate	% Extraction
December.....	1st	5.36	1.68	220.0	69.19
	2nd	6.56	1.01	206.0	85.02
	3rd	7.10	1.03	289.0	85.80
	4th	7.55	1.15	443.0	84.98
January.....	1st	6.65	1.17	568.0	82.58
	2nd	7.30	1.20	436.0	83.80
	3rd	5.70	1.00	364.0	82.68
	4th	4.70	0.90	335.0	82.95
Average.....		6.37	1.14	357.6	82.13

Due to the present excessive marketing charges, it is economy to sacrifice recovery to some extent, in order to produce a high-grade concentrate.

With the exception of the Buffalo Mines, the companies producing flotation concentrate depend on the smelters in the United States for the disposal of their products, and the schedules offered are not particularly attractive. A representative schedule stipulates a $12.00 to $14.00 treatment charge per ton and payment on New York quotation of 95% of the silver content. With silver selling at 70 cents an ounce the marketing cost per ton on a 300-ounce concentrate would be approximately as follows:—

Silver loss 5%—15 ounces @.70	$10.50
Treatment	14.00
Freight	11.32
Representation at the smelter	.50
Bagging and miscellaneous	2.00
Total marketing cost for one ton of concentrate	$38.32

This excessive marketing charge, representing 20% of the gross value of the product has led to considerable research work by the metallurgists of the district, to devise a satisfactory method for the local treatment of such concentrate.

Cyanidation, and wet desulphurization followed by cyanidation on the raw product, have not given much promise of success. Likewise, treatment by an oxidizing roast, and the cyanidation of the roasted product, has not succeeded.

A chloridizing roast followed by leaching, either with cyanide or an acid-brine solution has given the best results to date. In fact the Buffalo Mines is now treating 5 to 6 tons of flotation concentrate daily by a chloridizing roast, followed by an acid leach to extract the base metals, and then by a cyanide leach to recover the silver chloride. A 95% to 98% extraction of the silver values is being obtained by this method.

No attempt, so far, has been made to determine costs, but since the metallurgy of the treatment has been solved, no particular difficulty is anticipated in putting the process on a commercial basis. Of the machines considered to perform the chloridizing roast, the Holt-Dern furnace offers several advantages over furnaces of the ordinary reverberatory or McDougal type. This furnace is very simple in construction and cheap in operation. The standard furnace consists of a rectangular roasting chamber 7′ by 9′ by 4½′ deep with a hoppered bin under the grates for the discharged product. The fuel for the roast is furnished by the sulphur in the charge, and the roast is maintained by an air blast of 2,000 cubic feet per minute and 12 to 16 ounces pressure per square inch. Products to be roasted, with the requisite amount of salt, additional sulphur, (if necessary, in the form of pyritic ore) are all intermittently mixed, and then moistened to the proper degree. Assuming the furnace in running order, the cycle of operations is about as follows:—With the roasting zone within about two feet below the top of the furnace chamber, the mixed charge equivalent to about 20 inches of depth, is dropped into the furnace and leveled off. The blast is turned on and the roasting zone again moves up through the charge at the rate of about one foot an hour. When the roasting zone has again reached the top of the charge, the blast is turned off

and the grates shaken until the charge is again lowered about 20 inches. This completes the cycle. The roasted product is withdrawn from the hopper bin underneath the grates and transported to the leaching department.

The procedure in the leaching of the chloridized product as practiced at the Buffalo Mines is described in detail by Mr. Dye in his paper presented at this meeting.

The use of an acid-brine solution to dissolve out the economic metals, as practiced at the Tintic Milling Company, Silver City, Utah, would seem to offer some economy over the use of an acid leach, followed by cyanidation, but this has not yet been demonstrated on a commercial scale on Cobalt concentrates.

With the present method of marketing flotation concentrate, the profit per ton of ore milled, by a gravity concentration and flotation plant over one using only gravity concentration is indicated by the following comparison:

COMPARISON OF PROFITS BETWEEN GRAVITY CONCENTRATION AND FLOTATION PLANT, AND GRAVITY CONCENTRATION PLANT TREATING 200 TONS PER DAY.

HEADING 30 OUNCES SILVER.

GRAVITY CONCENTRATION AND FLOTATION.			GRAVITY CONCENTRATION		
96.80% x 30 oz., 29.04 oz. @ 70 cents per ounce		$20.33	81.29% x 30 oz. 24.39 oz. @ 70 cents per ounce		$17.00
Working costs	$1.40		Working costs	$1.00	
Interest and depreciation, royalties, etc., on additional plant	.05				
Marketing costs	$1.79	$3.24	Marketing costs	$1.27	$2.27
Profit		$17.09	Profit		$14.73

Profit from gravity concentration and flotation plant	$17.09
Profit from gravity concentration alone	14.73
Profit in favour of gravity concentration and flotation plant per ton of ore treated	2.36

The marketing costs are arrived at as follows:—

GRAVITY CONCENTRATION AND FLOTATION.		GRAVITY CONCENTRATION	
1 ton jig concentrate @ $68.30	$68.30	1 ton jig concentrate @ $68.30	$68.30
2.58 tons table concentrate @ $51.85	133.77	2.58 tons table concentrate @ $51.85	133.77
4.06 tons flotation concentrate @ $38.32	155.58	1.46 tons slime table concentrate @ $34.82	50.84
Total marketing cost on products from 200 tons of ore	$357.65		$252.91
Marketing cost per ton of ore milled	$ 1.79		$ 1.27

From these figures, the marketing cost of concentrate produced represents, in the case of gravity concentration, and flotation plant, 50% of the total cost of production, and the marketing cost of flotation concentrate alone represents approximately 20% of the total cost, so that any material reduction in this cost by reason of a local treatment plant for this product would result in a considerable saving which would be credited to flotation.

Some research work has been done to determine the amenability to recovery by flotation of the particular silver minerals. that occur in this district. These tests were carried out by mixing finely ground picked specimens of the particular mineral with a practically barren gangue, and floating the mixture. The tabulated results are given below.

TESTS TO DETERMINE AMENABILITY TO RECOVER BY FLOTATION OF ARGENTITE, PYRARGYRITE, PROUSTITE, METALLIC SILVER, FRIESLEBENITE AND MATILDITE.

Test No. 1.

	Formulæ	Head Assay Ozs. Ag	Tail Assay Ozs. Ag	Concs. Assay Ozs. Ag	Extraction
Argentite	Ag_2S	24.12	3.11	150.46	88.90%
Proustite	Ag_3AsS_3	11.96	2.42	215.44	80.68%
Pyrargyrite	Ag_3SbS_3	16.26	2.34	195.50	87.60%

Test No. 2.

	Formulæ	Head Assay Ozs. Ag	Tail Assay Ozs. Ag	Concs. Assay Ozs. Ag	Extraction
Argentite....	Ag_2S	23.30	5.60	469.3	76.88%
Proustite....	Ag_3AsS_3	12.60	2.70	604.4	78.92%
Pyrargyrite ..	Ag_3SbS_3	18.50	2.80	738.4	85.19%
Metallic silver	Ag	23.60	2.10	255.4	91.86%

Test No. 3.

Metallic silver	Ag	36.8	0.90	518.6	97.72%
Frieslebenite .	$(PbAg_2)_5Sb_4S_{11}$	9.2	2.9	416.5	68.96%
Matildite....	$Ag_2S.Bi_2S_3$	19.56	1.50	399.4	92.68%

The gangue material used for Test No. 1 was Nipissing cyanide tails. For Test No. *2*, ordinary river sand containing considerable iron oxide, and for Test No. *3*, washed river sand.

As shown by these tests, these minerals without exception are recoverable by flotation; and metallic silver, when finely ground, is particularly amenable to recovery by this process.

The adoption of flotation by the Cobalt mill operators has been rapid, and there are now only about 790 tons per day of current tailing that are not being treated by flotation.

By reason of its better recovery, and lower operating costs, even with the present excessive charge for marketing the concentrate, flotation shows a profit over cyanidation. When taken in connection with a local treatment plant for this concentrate, this profit will be more marked and flotation will eventually replace cyanidation on Cobalt ores.

There are approximately 2,500,000 tons of impounded tailing in the district averaging 4 to 6 oz. silver that can and undoubtedly will be treated by flotation at a substantial profit.

As the ores of the district decrease in grade, necessitating a cheap process with high recoveries, flotation will become increasingly important.

CANADIAN WOOD OILS FOR ORE FLOTATION.[1]

(Preliminary Paper).

[2]By C. S. Parsons and R. E. Gilmore.

Annual Meeting, Montreal, March, 1917.

Introduction.—The Mines Branch and the Forest Products Laboratories of the Dominion Government were asked to undertake a co-operative investigation of the possibility of obtaining a supply of flotation oils from Canadian woods.[3]

At the present time the entire supply of pine oils consumed in America is produced in the Southern States. So great has been the demand for these oils, owing to the increasing use of the flotation process for the concentration of ores, that the market has been depleted. Furthermore much of the so-called pine oil now being produced contains less than 20% pure pine oil. The price has more than doubled and is still increasing.

The difficulty then of obtaining pure pine oil for flotation purposes is by no means inconsiderable. The aim is, therefore, to produce in Canada a pine oil, or a substitute for pine oil, to meet this shortage and also render consumers in this country independent of United States sources of supply.

Acknowledgment is due Mr. A. A. Cole, President of the Institute, for instigating this investigation and to Dr. J. S. Bates, Superintendent of the Forest Products Laboratories, Montreal, for his zealous efforts in planning and directing the course of the work throughout.

Working Plan.—The working plan of the investigation as

---

[1]By permission of the Director, Mines Branch.

[2]C. S. Parsons, Mines Branch, Ottawa.

R. E. Gilmore, Forest Products Laboratories, Montreal.

[3]Since presenting this paper commercial tests have been conducted at Cobalt. These confirm the results of our laboratory experiments indicating that the hardwood creosotes can be employed as frothing agents in place of pine oil in the treatment of the Cobalt ores.

outlined by the Forest Products Laboratories is somewhat as follows:—

(1) To find a substitute for pine oil for which the mining companies are dependent for the supply, which is a limited one, from the United States.

(a) Investigation of oils, chiefly ketone and creosote oils from the destructive distillation of hardwood, for the purpose of finding a frothing oil suitable for use in ore flotation.

(b) Providing no satisfactory results were obtained from hardwood oils, to investigate the commercial possibilities of obtaining a pine oil from Canadian resinous woods.

(2) To utilize in place of coal tar and coal tar creosote the oils now produced by the destructive distillation of hardwood and which at present are of little market value.

Types of these oils are as follows:

Tar oils.....	heavy acid oil hardwood tar
Creosote oils	light creosote oils heavy creosote oils
Ketone oils..	heavy ketone oil ketone residue

Sources of supply.—The resinous woods available are red pine stumps, western yellow pine of British Columbia, and possibly white pine stumps.

It is well known that very large quantities of pine stumps exist throughout western and northern Ontario; whether or not these stumps can be collected and distilled on a commercial basis is a matter which requires more careful investigation, and one on which we are at present not able to advise.

There is, however, a large supply of hardwood oil available. The total amount of hardwood distilled per day in Canada is over 500 cords. The largest single producing plant is that of the Standard Chemical Co. at Longford Mills, which has a

capacity of 80 cords per day. The yield of oils[4] per cord is approximately:

Oil	[5]gals. per cord	gals. per day for Canada
[6]FPL No. 25, light creosote.....	1.600	800
FPL " 26, heavy "	0.486	243
FPL " 27, acid oil.....	0.284	140
FPL " 30, hardwood tar.....	16.00	8,000
FPL " 28, heavy ketone.....	2,000 per mont at Longford plant only.	
FPL " 29, ketone residue.....	5,000 per month at Longford plant only.	

FLOTATION TESTS.

The ore chosen for the first series of tests was the slime from the Coniagas mill, Cobalt, Ontario. This slime contains 5.7 ounces of silver per ton, and should be a fairly representative flotation ore for the silver area. A screen analysis of the slime is given below. It is a noteworthy fact that 62 per cent of the silver is contained in the 200 – mesh product.

SCREEN ANALYSIS

Sample of silver slimes from Coniagas Mines, Limited, Cobalt, Ont.

I. M. M. Standard Screens

Mesh.	App. Inches	Wt. Total	% Total	% Accumulative	Ozs. Ag.	% Total Ag.	Accumulative %
+ 90	.0055	79	7.9	.....	2.08	2.75	2.75
+100	.0050	52	5.2	13.1	2.28	1.99	4.75
+120	.0042	371	37.1	50.2	3.68	22.9	27.65
+150	.0032	46	4.6	54.5	4.48	3.46	31.1
+200	.0025	83	8.3	63.1	4.96	6.90	38.00
– 200	.....	369	36.9	100.0	10.00	62.00	100.00
Total....	.....	1000	.....	.....	5.96	100.00	.....

[4]For description of oils tested see final paper, page 58 *et seq.* this volume.
[5]Imperial gallons.
[6]Forest Products Laboratories, Montreal.

Average chemical composition of city tap water:

Calcium	carbonate..... 17	ppm.
"	sulphate..... 5	"
"	chloride..... 2.5	"
Magnesium	carbonate..... 8.0	"
Sodium	chloride..... 2.5	"
Magnesium	sulphate..... 3.0	"

Colour (40 p.p.m.) is almost entirely colloidal in nature, and carries a negative electrostatic charge.

In order to collect information covering the utilization of as many oils as possible it was decided, for the present at least, to confine the work to the one standard ore. Later the experiments will be continued to include a copper ore, such as the Bruce mine ore, and probably a zinc-lead ore from British Columbia.

The Janney testing machine was used throughout this experimental work. This apparatus consists of a cylindrical agitation chamber, fitted with baffles, and a spitzkasten, or settling chamber, in which the froth collects. The agitation is obtained by an impeller running at 1,800 r.p.m. A detailed description of this machine was published in the *Mining and Scientific Press*, for January 1st, 1916.

In conducting these experiments no special effort was made to produce a high-grade concentrate or to obtain high extraction. The aim was rather to produce comparative results; hence those familiar with the flotation of Cobalt ores will notice that results given in the following tables are not as good as obtained in practice. The method of procedure was to keep all conditions as nearly constant as practicable. The ore with the measured quantity of oil was placed in a pebble mill, and a pulp was made up of one part of water to one of ore. This charge was then ground for twenty minutes in order to break up the coal tar and thoroughly incorporate the oil and ore. The ground pulp was then poured into the flotation machine, and agitated for 6 minutes.

The temperature in each test varied between 5° C. and 15° C. The best results were obtained with the colder pulp. A rougher tailing and a rougher concentrate *only* were produced. The city tap water was used in all these tests, an analysis of which is given below.

The Results.—The accompanying tables 1-14 contain the data indicating the results obtained from the experiments conducted on the various oils. The assay of the concentrate and tailing is given in ounces per ton, and the weight of the concentrate in grams. The extraction is obtained from these figures. The total oil as well as any other reagent is given in pounds per ton of dry slimes. A separate table has been compiled for each oil, and the number of the oil is placed at the top of the table. The oil mixture in which the oil used is shown in the second column.

Table No. 1 contains the results obtained with a pine oil mixture. This mixture of 10% pine oil, 10% coal tar, and 80% coal tar creosote is used commercially at Cobalt, and has been found to give very satisfactory results. It has been adopted as the standard mixture for comparison in these tests. Test No. 1 shows the average result obtained. Test No. 2 the effect of adding caustic soda and it will be noted that there is a decided increase both in the extraction and grade of concentrate. This was found to be the case whenever caustic soda was added, irrespective of the oil used. Test No. 5 represents the best results obtained with this mixture.

TABLE 1

Oil No. 1.

Standard pine oil G.N.S. 55.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents.	Froth		Ext.	Tail ag.	Remarks
				Wt.	Ag.			
(1)	10% FPL No. 1; 10% coal tar; 80% coal tar creosote.	1.5		32	57.3	67.5	1.90	Very heavy froth, small bubbles.
(2)	10% FPL No. 1; 10% coal tar; 80% coal tar creosote.	1.5	4.0 lbs. NaOH	21	105	81.2	1.14	Not as heavy a froth as in test No. 1 slime was defloculated.
(3)	10% FPL No. 1; 10% coal tar; 80% coal tar creosote.	1.5	4.0 lbs. NaC1	24	70.1	62.7	2.06	Good froth.
(4)	10% FPL No. 1; 10% coal tar; 80% coal tar creosote.	1.0		36	39.0	53.0	2.60	
(5)	10% FPL No. 1; 10% coal tar; 80% coal tar creosote.	1.5		24	84.1	74.6	1.44	This is the best result obtained.

Tables Nos. 2 and 3 show respectively the results obtained with crude pine distillation from southern pine and crude distillate from red pine stumps. The flotation values of these two oils are identical on this ore.

TABLE 2

Oil No. FPL 4.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents.	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
(6)	Alone.	1.5		65	15.2		4.00	No selective action. P.T.T.Co. No. 350.
(7)	25% FPL No. 4; 7.5% coal tar; 67.5% coal tar creosote.	1.5		45	30.0	47.5	2.65	

TABLE 3

Oil No. FPL 14.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents.	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
8	Alone.	1.5		66	17.9		4.00	Very heavy froth like FPL No. 4.
9	10% FPL No. 14; 10% coal tar; 80% coal tar creosote.	1.5		12	104.5	42	3.56	1″ inch of froth

Tables Nos. 4, 5, and 6 show the results with light hardwood creosote, heavy hardwood creosote, and acid oil. It will be seen that these oils give equally as satisfactory results as pine oil. The highest extraction was obtained with the heavy hardwood creosote oil FPL No. 26. The froth was of a good character and broke down readily. The same surprising result was again obtained when the pulp was made alkaline with caustic soda. (See test No. 21.)

TABLE 4

Oil No. FPL 25.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents.	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
10	10% FPL 25; 10% coal tar; 80% coal tar creosote.	1.5		6	214.0	46.4	3.0	Only a little froth formed.
11	28.6% FPL 25; 10% coal tar; 71.4% coal tar creosote.	1.5		21	77.7	67.5	1.64	
12	40% FPL No. 25; 6% coal tar; 54% coal tar creosote.	1.5		21.5	83.3	65.0	2.00	Good heavy froth.
13	40% FPL No. 25; 60% pine tar P.T.T. Co. No. 750.	1.5		51	20.4	38.0	3.78	All froth.
14	40% FPL No. 25; 60% coal tar. G.N.S. No. 28.	1.5		28	52.0	55.6	2.46	Good heavy froth. Tar separated out.

TABLE 5

Oil No. FPL 26.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents.	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
15	40% FPL No. 26; 6% coal tar; coal tar creosote 54%.	1.5		33	70.7	77.4	1.46	This is the best result obtained
16	40% FPL No. 26; coal tar 60%.	1.5		28	48.0	51.0	2.70	Good froth.
17	40% FPL No. 26; coal tar 6%; coal tar creosote 54%.	1.5		29.5	63.3	64.5	2.13	Good froth, an average result.
18	40% FPL No. 26; coal tar 30%; coal tar creosote 30%.	1.5		30	60.5	64.0	2.10	Average result obtained.
19	40% FPL No. 26; coal tar 30%; coal tar creosote 30%.	1.75		30	70.3	74.0	1.54	
20	50% FPL No. 26; coal tar 5%; coal tar creosote 45%.	1.5		39	34.2	50.0	2.96	Very heavy froth.
21	30% FPL No. 26; coal tar 7%; coal tar creosote 63%.	1.5	4.0 lbs. NaOH	7	222.5	63.0	1.8	All the froth was not drawn.

TABLE 6

Oil No. FPL 27.

Weight of ore 500 grams. Assay 5.6 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
22	40% FPL No. 27; coal tar 6%; coal tar creosote 54%.	1.5		15	139.0	70.5	1.8	Only one test run on this oil.

Table No. 7. This is a ketone oil from the manufacture of acetone; the frothing properties of which are not very pronounced. The best results were obtained in a pulp made alkaline with caustic soda. When used in a neutral pulp the extraction was very low. The average of a number of experiments is given in test No. 24. Test No. 25 shows the best results obtained with this oil.

TABLE 7

Oil No. FPL 28.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
23	10% FPL No. 28; coal tar 10%; coal tar creosote 80%.	1.5		13	109.0	49.0	3.02	Very little froth.
24	26% FPL No. 28; coal tar 7.4%; creosote 66.6%.	2.0	4.0 lbs. NaOH	10	235.0	79.0	1.3	Average result. Slimes were deflocculated.
25	26% FPL No. 28; coal tar 7.4%; creosote 66.6%.	2.0	4.0 lbs. NaOH	4.5	474.7	73.5	1.56	Best result obtained.
26	40% FPL No. 28; coal tar 6%; coal tar creosote 54%.	1.5		8	173.7	50.0	3.02	Poor frother.

Table No. 8. This oil is the mixture of hardwood creosotes in proportion as they are produced. The average of a number of experiments is represented in test No. 27. The collecting oils used were coal tar and coal tar creosote. In tests numbers 28, 29, and 30 the coal tar and coal tar creosote were replaced by hardwood tar distillate and ketone residue, and it is quite apparent that this

substitution was unsatisfactory. Both the hardwood tar and ketone residue distillates have good frothing properties.

TABLE 8

Oil No. FPL 31.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
27	30% FPL No. 31; coal tar 7%; coal tar creosote 63%.	1.5		20	87.7	61.0	2.34	Good heavy froth.
28	40% FPL No. 31; coal tar 10%; Hardwood tar distillate No. 30 A, 50%.	1.25		47	30.0	53.7	2.96	Too much froth.
29	50% FPL 31; hardwood tar distillate No. 30 A. 50%.	1.0		52	25.3	52.0	2.72	Too much froth.
30	50% FPL 31; ketone residue No. 29, 50%.	1.0		52	30.3	59.0	2.42	Too much froth.

Table No. 9 contains the results from oil FPL No. 32, which is composed of the lighter oils distilling below 230° C. from red pine stumps, such as turpentine and pine oils. The tests prove it to be inferior to pine oil FPL No. 1. (G.N.S. No. 55).

TABLE 9

Oil No. FPL 32.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
31	10% FPL 32; coal tar creosote 80%; coal tar 10%.	1.5		Scum			3.92	Just a thin scum was formed.
32	10% FPL 32; 80% coal tar creosote; 10% coal tar.	2.0		6	252.0	52.6	2.76	

Table No. 10 shows the results from oil FPL No. 33, which as mentioned above, is the rosin and tar oil distilling between 230° C. and 345° C. from red pine stumps.

TABLE 10

Oil No. FPL 33.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
33	10% FPL No. 33; coal tar 10%; coal tar creosote 80%.	1.5		Scum		23.0	4.46	No froth.

Table No. 11 gives the results obtained from using a pine oil distilled from red pine stumps. In spite of the fact that double the quantity is required, it is gratifying to note that this pine oil produces higher results than steam distilled pine oil obtained from southern pine.

TABLE 11

Oil No. FPL 35.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
34	10% FPL No. 35; coal tar 10%; coal tar creosote 80%.	1.5		6	190.2	40.5	3.4	½" of froth
35	10% FPL No. 35; coal tar 10%; coal tar creosote 80%.	1.5	4.0 lbs. NaOH	9	234.8	74.0	1.52	Slimes were deflocculated.
36	20% FPL No. 35; coal tar 8%; coal tar creosote 72%.	1.5		32	63.3	68.7	2.0	Good heavy froth.
37	20% FPL No. 35; coal tar 8%; coal tar creosote 72%.	2.0		35	59.6	74.0	1.56	Good heavy froth.

Table No. 12. This table shows the results of experiments with a hardwood creosote supplied by the General Naval Stores Co. The result was not quite as good as with the Standard Chemical Iron and Lumber Company's heavy creosote.

TABLE 12

Oil No. FPL 38.

Weight of ore 500 grams. Assay 5.7 oz. Ag.

Test No.	Oil mixture used	Total oil lbs.	Reagents	Froth		Ext.	Tail Ag.	Remarks
				Wt.	Ag.			
38	40% FPL No. 38; coal tar 6%; coal tar creosote 54%.	1.75		27	65.3	65.0	2.0	Good froth.

Table No. 13. This table contains the data of two tests conducted on a copper ore containing chalcopyrite in a clean white quartz gangue. Two oil mixtures were made up, one containing 25% pine oil G.N.S. No. 55; 65% coal tar creosote and 10% coal tar and the other 40% heavy hardwood creosote; 54% coal tar creosote and 6% coal tar. The highest extraction and the cleanest concentrate was obtained with the hardwood creosote mixture. These experiments were inserted to illustrate the general utility of the hardwood creosotes and that in numerous cases they could be used to advantage as a substitute for pine oil.

TABLE 13

Weight of ore 500 grams. Assay 7.9 copper.

Test No.	Oil mixture used	Total oil lbs.	Froth		Ext.	Tails % Cu	Remarks
			Wt.	Cu %			
1	25% pine oil FPL No. 1; 65% coal tar creosote; 10% coal tar.	1.5	155.	23.4	91.8	0.94	
2	40% Hardwood creosote FPL No. 26; 54% coal tar creosote 6% coal tar.	1.5	157.	23.6	93.1	0.80	

Table No. 14 was compiled to illustrate the beneficial effect of adding caustic soda to the flotation pulp.

TABLE 14

Showing effect of NaOH.

Test No.	Oil mixture used	Total oil	Total NaON	Froth		Ext.	Tails Ag.
				Wt.	Ag.		
2	10% Pine oil; 10% coal tar; 80% coal tar creosote.	1.5	4	21	105.	81.2	1.14
1	10% Pine oil; 10% coal tar; 80% coal tar creosote.	1.5		32	57.3	67.5	1.90
	Heavy hardwood creosote						
21	30% FPL No. 26; 7% coal tar; 63% coal tar creosote.	1.5	4	7	222.5	63.0	1.80
18	40% FPL No. 26; 6% coal tar; 54% coal tar creosote.	1.5		30	60.5	64.0	2.10
	Ketone Oil						
25	26% FPL No. 28; 7.4% coal tar; 66.6% C. T. creosote.	2.0	4	4.5	474.7	73.5	1.56
26	40% FPL No. 28; 6% coal tar; 54% coal tar creosote.	1.5		8	173.7	50.0	3.02
	Pine oil, red pine stumps.						
35	10% FPL No. 35; 10% coal tar; 80% coal tar creosote.	1.5	4	9	234.8	74.0	1.52
37	20% FPL No. 35; 8% coal tar; 72% coal tar creosote.	2.0		35	59.6	74.0	.1.56

The great improvement in both the grade of the concentrate and in the extraction warrants investigation from a practical and commercial standpoint. It is probable that the amount of

caustic soda could be reduced in practice. The caustic soda apparently deflocculates the slime, releasing any silver values enclosed in the floccules of gangue. In the absence of caustic soda, the oil must penetrate the floccules of slime in order to reach and oil the mineral particles.

(Final Paper)

In August, 1916 the Canadian Government at the request of the Cobalt mining interests began active work on an investigation of Canadian wood oils for ore flotation.

The success of the process for the concentration of the low grade silver ores had already been demonstrated. On account of the difficulty which the flotation plants, (the number of which was rapidly growing) experienced in obtaining an adequate supply of imported pine oil, it was decided to investigate the possibilities of either finding a substitute for pine oil, which was found necessary as a selective frothing oil, or of producing a satisfactory grade of pine oil in sufficient quantities from Canadian resinous wood waste.

The collecting of the wood oils was carefully planned and speedily carried out by the Forest Products Laboratories, Montreal, where physical and chemical examinations were made. The flotation tests were made by the Division of Ore Dressing and Metallurgy, Mines Branch, Ottawa.

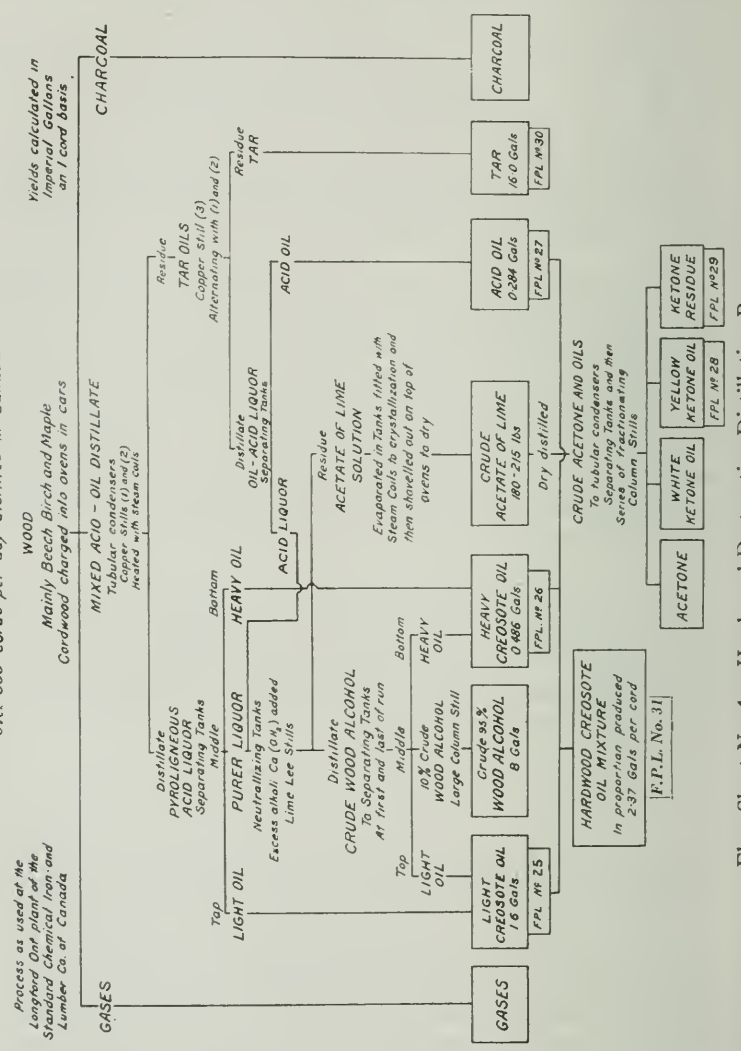
Part 1.

The Collection and Examination of the Wood Oils.

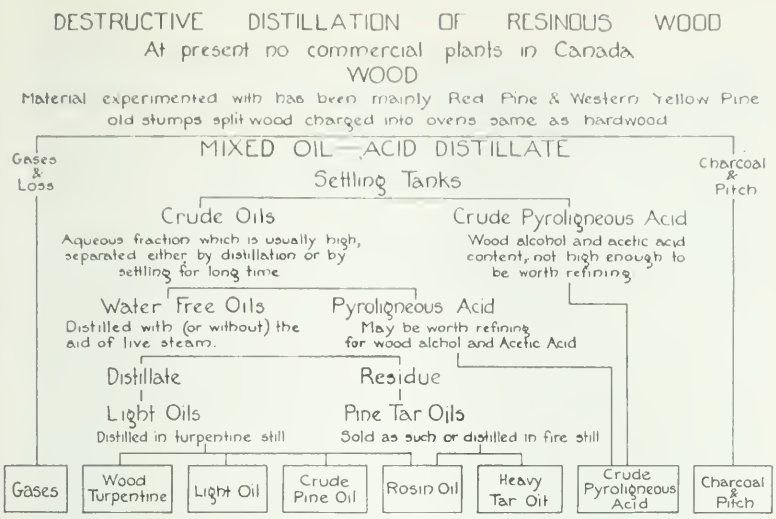
Experience has shown that the flotation value of an oil or oil mixture for a given ore, and especially the silver ores of the Cobalt area, cannot as yet be forecasted. Hence it was necessary to collect all available wood oils, both resinous and hardwood, of which the price seemed to be reasonable; and in the collecting

of these oils special care was taken to learn their origin and to make sure, as far as possible, that these oils could be duplicated commercially.

With the permission of the Standard Chemical, Iron and Lumber Company of Canada, Limited, the accompanying flow-



Flow Sheet No. 1.—Hardwood Destructive Distillation Process.


DESTRUCTIVE DISTILLATION OF RESINOUS WOOD
At present no commercial plants in Canada
WOOD
Material experimented with has been mainly Red Pine & Western Yellow Pine
old stumps split wood charged into ovens same as hardwood
MIXED OIL—ACID DISTILLATE
Settling Tanks
Gases & Loss
Charcoal & Pitch
Crude Oils
Aqueous fraction which is usually high, separated either by distillation or by settling for long time
Crude Pyroligneous Acid
Wood alcohol and acetic acid content, not high enough to be worth refining
Water Free Oils
Distilled with (or without) the aid of live steam.
Pyroligneous Acid
May be worth refining for wood alchol and Acetic Acid
Distillate
Residue
Light Oils
Distilled in turpentine still
Pine Tar Oils
Sold as such or distilled in fire still
Gases
Wood Turpentine
Light Oil
Crude Pine Oil
Rosin Oil
Heavy Tar Oil
Crude Pyroligneous Acid
Charcoal & Pitch

Flow Sheet No. 2.—Resinous Wood Destructive Distillation Process.

sheet (No. 1) outlines the origin of the hardwood oil samples collected from their Longford, (Ontario) plant.

As yet there is no commercial source of resinous wood oils in Canada. The purest grade of pine oil is derived by the steam and solvent process, and a less pure grade by destructive distillation, from the longleaf southern pine of the Southern States. As it is very doubtful that any process other than destructive distillation will prove successful for Canadian resinous wood waste a flow sheet (No. 2) of the destructive distillation process only is given here. Resinous wood oil samples collected and tested for flotation value include crude oil distillate and its fractions, from experimental runs made on red pine stumps at the Buffalo Mines, Cobalt, Ontario, and crude oil distillate including crude turps, crude light oil and pine tar oil from experimental distillation runs, by the Pritchard heat control process, on British Columbia western yellow pine stumps at Wilmington, North Carolina.

In addition to a knowledge of the origin of an individual oil, a fractional distillation and determination of physical properties of the fractions in detail were considered ample in order to describe the oil and to be able to obtain a duplicate sample when desired. In the following standard method adopted for the exam-

ination of the oils by fractional distillation the description of an oil by curves was employed. A few of the more successful oils are illustrated in Figures 1, 2, 3 and 4.

Details of this standard fractional distillation method are as follows:

Distillation Flask Used:—Ordinary half litre Engler distillation flask made of Pyrex glass.

Fractionating Column:—Two inches of standard glass ring beads supported in platinum or aluminum wire basket resting on hollow glass tubing; the upper end of the glass tubing is bent in the loop and fits the lower end of the stem of the flask, while the lower end rests on the bottom of the distillation flask itself. This glass tubing support with the upper loop end drawn out and sealed and the lower end left open also helps to prevent bumping, by causing a steady stream of fine bubbles of the expanding air within the tube to escape from the contact point at the lower end as the temperature rises.

Condenser:—A short Liebeg water condenser held in position at an angle of about 45 degrees and fitted so that for a high boiling oil the water could be run out from the jacket and a stream of air passed through either by pressure or suction.

Source of Heat:—An electric oven was constructed for the purpose in which the distilling flask was set, leaving only the neck of the flask with its fractionating column of beads exposed. The rate of distillation, which it is important to keep uniform, is controlled by varying the electric current for different oils. Thus the heat increases with the current in the heat equation. Heat = Constant $\times$ RC^2T, where R = resistance in ohms, C = current in amperes and T = time in seconds.

Thermometer:—A 360-degree mercury thermometer was inserted so that the top of the mercury bulb was level with the top of the delivery tube opening. The standard correction for emergent stem was added, viz.: $N\ (t\text{-}t^1)\ 0.000143$, where N = number of degrees on the part of the mercury column outside the bath, t = temperature read and t^1 = the average temperature of the air surrounding the emergent part of the stem.

Rate of Distillation:—The average rate of distillation was 2.0 to 2.2 grams per minute, running between 90 and 120 drops in the same time, depending on the density of the distillate.

Amount of Sample Used and Fractional Cuts:—350 grams was the usual sample taken and 10% fractions (about 35 grams) were separated, the temperature and time at each cut being noted. Each fraction was collected in a small cylinder stand ¾ inches in diameter and 5 inches high, suitable for taking specific gravity readings.

Specific Gravity and Refractive Index Readings:—Specific gravity readings were taken with a Westphal balance and at room temperature, while the refractive index readings were observed by an Abb refractometer, also at room temperature. All readings were corrected to 15°C; in the case of specific gravity 0.0008 was added to the observed reading for every degree above 15°C, and, in the case of refractive index, 0.000365 was added to the observed reading for every degree above 15°C.

It is evident that in order to check one distillation with another on the same sample a uniform rate of distillation is all important, and a uniform rate of distillation is directly dependent on a uniform rate of heating. Although the best checks were obtained when using an electric oven, remarkably close checks were obtainable with the distilling flask set in a sand bath over a gas flame, which flame could be regulated by a screw pinch cock. This is true and adaptable for oils with boiling point range (say) below 250°C. For higher boiling oils, especially tar oils, a free flame is necessary, but a fractionating column of beads is inadvisable. When using a gas flame, an asbestos box or better an asbestos cloth cover fitted over and around the distillation flask served to give better heat control and resembled more closely the conditions when the electric oven was used.

Although not necessary when a uniform rate of heating is supplied, it was found even for the crude hardwood creosote oils that a fractionating column of beads aided in giving a more uniform rate of distillation; the curves of distillations where beads were used showed more pronounced deflection points and gave much better checks than when no beads were employed.

Determination of Water Content of an Oil Sample:—Where the moisture content of an oil is less than 10% and where only a small fraction of the oil distils below say 150°C, the aqueous fraction will be practically all in the first fractional cut and if collected in a small graduated cylinder can be read off directly. When an oil sample has a high aqueous content it is best first to run a preliminary distillation up to say 200°C and then after noting and separating the water layer return the oil content to the cooled distilling flask and begin again. A few pieces of broken porous porcelain or pumice stone and a small amount of paraffin wax put into the flask with the oil, generally aid greatly in preventing superheating, bumping and excessive frothing when distilling an oil sample containing more or less water.

Oils for Flotation Tests:—1st. Among the first series of oils tested were the original crude oils as collected. These are represented by serial numbers.

2nd. A sample of the total distillate, being the combined fractions from a distillation run, was in each case saved and tried out to see if the removal of free carbon and heavy pitch or coke residue from the oil improved its flotation value. These distillates are represented by an alphabetic letter after the number of oil from which it was derived.

3rd. Individual fractions and groups of fractions were saved for the purpose of determining just what part of a crude oil was of most flotation value.

4th. Preliminary chemical separations of the hardwood creosote oils into neutral and acidic (phenolic and creosotic) constituents were made and the respective flotation values of these two group separates tested.

A description of all the oils submitted for flotation tests is given in Table I. A more detailed description of each of these oils is available at the Forest Products Laboratories where also duplicate samples are in reserve.

TABLE 1.—DETAILS OF THE WOOD OILS COLLECTED AND EXAMINED

F.P.L.[1] No.	Description of Oil	Appearance	Sp. Gr. and Ref. Index at 15° C	Flotation action on Cobalt ores
1	*Pine Oil* G.N.S.[2] No. 5 (same as G.N.S. No. 55 from longleaf southern pine by the steam and solvent process. 95% by wt. distilled between 190° C and 220° C where sp. gr. was 0.890 to 0.946 and ref. index 1.473 to 1.488.	Clear, light amber	0.9330 1.4837	Excellent selective frother; poor collector
2	*Pine Oil* redistilled from G.N.S. No.'s 55 and 4. 100% distilled between 190° C and 220° C.	Clear, light amber	0.9310 1.4848	Same as above
3	*Pine Oil* G.N.S. No. 6 (same as G.N.S. No. 4) from longleaf southern pine by destructive distillation process. 67% distilled between 190° C and 220° C where sp. gr. range was 0.890 to 0.951. Only 90% distilled below 246° C.	Clear, amber	0.934 1.4995	Good selective frother; poor collector
4	*Crude Oil Distillate* P.T.T.[3] No. 350 from southern pine by destructive distillation. 50% by wt. only, distilled below 345° C. On primary distillation only 13% distilled between 190° C and 245° C where sp. gr. range was 0.897 to 0.965.	Black opaque	1.0190	Good frother but non-selective; poor collector
14	*Crude Oil Distillate* from red pine stumps. Yield of this oil is 13.9% by wt. of original wood or 36 Imp. gals. per 2,600 lb. cord. On secondary distillation only 5% of original oil distilled between gravities of 0.890 and 0.965; 28% of this oil is distillable with steam.	Black opaque	1.009	Good frother but non-selective; poor collector

[1] Forest Products Laboratories of Canada.
[2] General Naval Stores Company, New York.
[3] Pensacola Tar and Turpentine Co., Gull Point, Florida.

F.P.L. No.	Description of Oil	Appearance	Sp. Gr. and Ref. Index at 15° C	Flotation action on Cobalt ores
17	*Crude Turps* from western yellow pine stumps. Yield = 10.8 Imp. gals. per 2,600 lb. cord. Primary distillation gave over 60% distilling below 174° C with sp. gr. below 0.870.	Black, opaque	0.9070	Poor frother
18	*Crude Light Oil* from western yellow pine stumps. Yield = 2.4 Imp. gals. per 2,600 lb. cord. Primary distillation gave over 50% distilling below 190° C with sp. gr. below 0.88.	Black, opaque	0.9300	Good frother and fairly selective
19	*Crude Pine Tar Oil* from western yellow pine stumps. Yield = 38 Imp. gals. per 2,600 lb. cord. Less than 12% distilled below 246° C having gravity below 1.01. This oil yields 6.5% of crude pine oil suitable for flotation of Cobalt ore.	Black, opaque		Good frother but non-selective; poor collector
23	*Coal Tar* from the Dominion Tar and Chemical Co., Sault Ste. Marie, Ont.	Black, opaque		Excellent collector
24	*Coal Tar Creosote* from the same company as above. 80% by wt. distilled between 170° C and 325° C.	Black, opaque		Excellent collector
25	*Light Hardwood Creosote Oil* from top of separating tanks after crude pyroligneous acid copper stills and also after lime-lee stills. 35% by wt. distilled below 150° C having gravity below 0.885. Over 90% distilled below 255° C of which gravity was below 1.025.	Black, opaque	0.9550	Good frother and selective, fair collector
25A	*Light Hardwood Creosote* being 35% of No. 25. Distillation range 60°C—150°C Sp. Gr. " 0.880—0.882 Ref. Index " 1.407—1.458	Clear, light amber	0.8880 1.4288	Poor frother

F.P.L. No.	Description of Oil	Appearance	Sp. Gr. and Ref. Index at 15° C	Flotation action on Cobalt ores
25B	*Light Hardwood Creosote* being 57% of No. 25. Distillation range 150°C—255°C Sp. Gr. " 0.882—0.990 Ref. Index " 1.458—1.520	Clear, reddish amber	0.9750 1.5077	Good selective frother; good collector
26	*Heavy Hardwood Creosote Oil* from bottom of same separating tanks as above. Less than 24% including 8.5% aqueous distilled below 200° C. 62% distilled between 200° C and 260° C where sp. gr. range was 1.013 to 1.044.	Black, opaque	1.0375	Very good selective frother; good collector
26A	*Heavy Hardwood Creosote Distillate* being 82% of No. 26. Distillation range 60°C—260°C Sp. Gr. " 0.924—1.036 Ref. Index " 1.434—1.534 75% acidic, 23% neutral.	Reddish amber	1.0300 1.5115	Good selective frother and good collector
27	*Heavy Hardwood Creosote Oil* known as 'acid oil' coming from separating tanks after heavy oil and tar still. Less than 14% including 5.25% aqueous distilled below 200° C. 75.5% distilled between 200° C and 265° C where sp. gr. range was 1.0435 to 1.0675.	Black, opaque	1.0545	Very good selective frother; good collector
27A	*Heavy 'Acid Oil' Distillate* being 87.5% of No. 27. Distillation range 90°C—265°C Sp. Gr. " 1.023—1.063 Ref. Index " 1.479—1.530 90% acidic, 9% neutral.	Reddish amber	1.0490 1.5210	Same as No. 27.
28	*Heavy Yellow Ketone Oil* from the refining of acetone. Of the several ketone oils this sample is the heaviest of the series leaving ketone residue. Over 90% distilled below 120°C where sp. gr. range was 0.81 to 0.82 and ref. index 1.389 to 1.400.	Clear, amber	0.8275	Fair frother, selective action greatly increased by alkali (NaOH)

F.P.L. No.	Description of Oil	Appearance	Sp. Gr. and Ref. Index at 15° C	Flotation action on Cobalt ores
29	*Ketone Residue*, of thick syrupy nature being residue after the refining of acetone and various ketone oils. 70% is distillable below 300° C above which it is hard to proceed under ordinary conditions. Over 60% distilled between 210° C and 300° C where sp. gr. range was 0.91 to 1.014.	Black, opaque	1.0345	Good selective frother and fair collector
29A	*Ketone Residue Distillate* being 70% of No. 29. Distillation range 95° C—310° C Sp. Gr. " 0.910—1.020 Ref. Index " 1.470—1.590	Reddish brown, slightly opaque	0.943	Good frother and fairly selective
30	*Hardwood Tar* being residue left in tar stills after all available acid-alcohol liquor and creosote oils have been removed. Without the aid of a small amount of superheated steam it is difficult to distill more than 30% of this tar. 39% including 9% aqueous distilled below 300° C of which sp. gr. was below 1.132.	Black, opaque and quite viscous		Good frother, but non-selective
30A	*Hardwood Tar Distillate* being 30% of No. 30. Distillation range 90°C—235°C Sp. Gr. " 1.025—1.14 Ref. Index " 1.490—1.50 Roughly—50% acidic	Reddish brown, almost opaque	1.058 1.495	Good frother, poorly selective; and fair collector
31	*Hardwood Creosote Mixture* being mixture in proportion produced of No. 25—Light creosote oil No. 26—Heavy creosote oil and No. 27—Acid oil. See F.P.L. No. 31A.	Black, opaque	0.9985	Good selective frother and good collector
31A	*Hardwood Creosote Mixture Distillate* being 87% of No. 31. Distillation range 55°C—255°C Sp. Gr. " 0.882—1.022 Ref. Index " 1.416—1.524 Roughly—50% acidic, 50% neutral.	Clear, dark amber	0.989	Good frother and selective; fair collector

F.P.L. No.	Description of Oil	Appearance	Sp. Gr. and Ref. Index at 15° C	Flotation action on Cobalt ores
32	*Crude Light Oil* from red pine stumps being oils distilling below 235° C from F.P.L. No. 14 having sp. gr. below 0.995.	Dark amber	0.9293 1.4830	Poor frother in quantities used; poor collector
33	*Rosin and Tar Oil* from red pine stumps being oils distilling between 235° C and 345° C in primary distillation of F.P.L. No. 14 having sp. gr. range 0.995 to 1.017	Reddish amber	1.0150	Poor frother in quantities used.
34	*Turps and Light Oil* from secondary distillation of oil distillate from red pine stumps, being fraction of No. 32 distilling up to 75° C having gravity below 0.886 and refractive index 1.451 to 1.477.	Light amber	0.8900 1.4708	Poor frother in quantities used.
35	*Crude Pine Oil* from red pine stumps being fraction from distillation of No. 32, distilling between 175° C and 216° C (using beads as fractionating column) and having gravity range of 0.886 to 0.995 and Ref. Index range of 1.477 to 1.503. Yield = 4.3 Imp. gals. per 2,600 lb. cord.	Clear, light amber	0.9443 1.4942	Excellent frother and selective; fair collector
36	*Light Rosin Oil* from red pine stumps being fraction on No. 32 distilling between 216° C and 250° C having sp. gr. range 0.995 to 1.03 and ref. index 1.503 to 1.517.	Reddish amber		Good frother; poor selector
37	*Light Hardwood Creosote Oil* (G.N.S. No. 17). 50% of this oil distilled below 150° C with gravity range between 0.875 to 0.885. 90% distilled below 240° C with gravity below 1.002 leaving 10% hard pitch residue.	Black, opaque	0.9370	Good frother in quantities used; fair collector

F.P.L. No.	Description of Oil	Appearance	Sp. Gr. and Ref. Index at 15° C	Flotation action on Cobalt ores
37A	*Light Hardwood Creosote Oil* from primary distillation of No. 37. Distillation range 60°C—240°C Sp. Gr. " 0.87 —1.002 Ref. Index " 1.411—1.523	Clear, dark amber	0.9195 1.4665	Fair frother
38	*Heavy Hardwood Creosote Oil* (G.N.S. No. 18). 63% of this oil distilled below 240° C having gravity range 0.871 to 1.079 leaving 37% hard pitch residue.	Black, opaque	1.0620	Good frother; good selector; good collector
39	*Hardwood Oil Fraction 20% of No. 31* (hardwood creosote mixture). Distillation range 55°C—152° C Sp. Gr. " 0.880—0.890 Ref. Index " 1.410—1.440 28% acidic, 70% neutral.	Clear, light amber	0.8910 1.4297	Poor frother
40	*Hardwood Oil Fraction 18.5% of No. 31.* (Hardwood creosote mixture) Distillation range 152°C—202°C Sp. Gr. " 0.890—0.985 Ref. Index " 1.440—1.505 33.5% acidic; 66.5% neutral	Clear, light amber	0.9623 1.4883	Good frother; good selector; fair collector
41	*Hardwood Oil Fraction 19.5% of No. 31.* (Hardwood creosote mixture) Distillation range 202°C—220°C Sp. Gr. " 0.985—1.020 Ref. Index " 1.505—1.520 66% acidic; 34% neutral	Clear, amber	1.0142 1.5182	Good frother; good selector; fair collector
42	*Hardwood Oil Fraction 20% of No. 31.* (Hardwood creosote mixture) Distillation range 220°C—240°C Sp. Gr. " 1.020—1.025 Ref. Index " 1.520—1.525 69% acidic; 31% neutral.	Clear, dark amber	1.0230 1.5254	Good frother; good selector; fair collector

F.P.L. No.	Description of Oil	Appearance	Sp. Gr. and Ref. Index at 15° C	Flotation action on Cobalt ores
43	*Hardwood Oil Fraction 10% of No. 31.* (hardwood creosote mixture). Distillation range 240°C—255°C 50% acidic; 50% neutral.	Reddish amber	1.0143 1.5240	Good frother; fair selector
44	*Pine Oil* from western yellow pine (from crude turps No. 17) Distillation range 190°C—246°C Sp. Gr. " 0.880—0.9540 Ref. Index " 1.4820—1.508 Yield = 0.75 Imp. gals. per 2,600 lb. cord.	Clear, light amber	0.9320 1.4945	Good frother; comparing with No. 3
45	*Crude Pine Oil* from western yellow pine stumps obtained from crude light oil No. 18 and pine tar oil No. 19. Distillation range 190°C—245°C Sp. Gr. " 0.895—0.989 Yield = 0.35 and 2.6 Imp. gals. per 2,600 lb. cord from No. 18 and No. 19 respectively.	Dark amber	0.9480 1.4960	Same as above.
46	*Light Oils*—turps and light oil from western yellow pine stumps obtained from crude turps No. 17 and crude light oil No. 18.	White to amber	0.8735 1.4780	Very poor frother
47	*Neutral Oil — 50% of No. 31A* (hardwood creosote mixture) after extraction with dilute H_2SO_4 and NaOH.	Amber	0.9260 1.4900	Good frother; fairly selective

	% of No. 47	Dist. Range	Sp. Gr.	Ref; Index
47A	23.4	59°C—150°C	0.8805	1.4442
47B	25.4	150°C—200°C	0.9113	1.4805
47C	42.8	200°C—290°C	0.9585	1.5120

48	*Acidic Oil—50% of No. 31A* (hardwood creosote mixture—being portion extracted by dilute NaOH and then recovered by acidifying and distilling. Only about 1% distilled below 200°C. 36% dist. 200–215°C—sp. gr. 1.025 40% " 215–230°C— " " 1.050 23% " 230–255°C— " " 1.050	Dark amber	1.0425 1.5260	Good selective frother

F.P.L. No.	Description of Oil	Appearance	Sp. Gr. and Ref. Index at 15° C	Flotation action on Cobalt ores
48A	*Acidic Oil from No. 26A and No. 27A* (hardwood creosote oils) Distillation gave the following:— % by volume Corrected temperature: 26A 27A up to 200°C 10.0 7.5 200°C—215°C 31.0 19.5 215°C—230°C 31.0 37.5 230°C—255°C 28.0 35.5	Dark amber		Same as above
49	*Neutral Hardwood Oil from No. 26A* (heavy hardwood creosote oil) and from 27A (heavy acid oil). 18% of this oil distilled below 200°C	Dark amber	0.9634 1.5090	Same as No. 48
50	*Crude Wood Oil*—a by-product from top of digesters of the alkaline sulphate process for manufacturing paper pulp from Jack pine, spruce, etc. The crude oil is of grease-like consistency while its distillate is a light oil. Both have a very disagreeable odor. Only 11.8% distilled below 250°C. 69.1% by wt. distilled between 250°C and 130°C wheresp. gr. range was 0.920 to 0.935 and ref. index 1.480 to 1.522.—Over 19% remained above 310°C as a hard pitch residue.	Dark, yellow, thick emulsion	0.9820	Unsatisfactory

Pine Oil as a Standard Frothing Oil.—The purest grade of pine oil available (namely G.N.S. No. 5) was chosen as the standard frothing oil in this investigation. In the standard oil mixture adopted the pine oil was 10%, the balance being 80% coal tar creosote and 10% coal tar.

Practically all the pine oil of commerce is obtained from the longleaf southern pine. In this connection a definition of pine oil is interesting. According to specifications as adopted by the United States Navy Department, pine oil must be a properly prepared light straw coloured oil, produced by redistillation of heavy boiling point fractions resulting from the steam distillation of wood turpentines; have a strong aromatic odor resembling turpentine: specific gravity not greater than 0.937 and not less than 0.933 at 60°F. One hundred cubic centimeters of the oil

when subjected to distillation in a standard Engler distilling flask must yield at least 95% of distillate between the temperatures 375°F and 475°F (190.5°C - 246°C).

It is also interesting to note the difference between pine oil produced by the steam and solvent process and pine oil produced by destructive distillation. A good grade of each of these oils is supplied by the General Naval Stores Company, New York, the oils being known as G.N.S. No. 5 and G.N.S. No. 6 respectively. 95% by weight of the pine oil from the steam and solvent process distilled between 190°C and 220°C where the specific gravity range was from 0.890 to 0.946 and the refractive index range from 1.473 to 1.4888, while 67.0% by weight of pine oil from the destructive distillation process distilled between 190°C and 220°C and only 87.0% between 190°C and 246°C where the specific gravity range was from 0.890 to 0.955 and the refractive index range from 1.485 to 1.530. (See Figure 1).

As will be noticed this sample of pine oil from the destructive distillation of southern pine does not conform with the U.S. Navy Department specifications, while pine oil from the steam and solvent process comes well within the above specifications. Judging from actual flotation results, for Cobalt area ore the use of steam distilled pine oil is to be recommended greatly in preference to the destructive distillation grades, and graded specifications approaching 95% distilling between 190°C and 220°C with the proper specific gravity range seem worthy of consideration in buying pine oil for flotation purposes. A pure pine oil is not always necessary for flotation purposes, but in using destructive distillation grades their value appears to be in direct proportion to the pine oil content as defined above.

Pine Oil from Canadian Resinous Woods:—In Canada, the most promising resinous wood materials are the red (Norway) pine stumps of Northern Ontario and the western yellow pine stumps of British Columbia. The old stumps, after the tree has been felled from 10 to 20 years, in many cases have sound hearts rich in oleoresin. Although the resin content is high, the yield of wood turpentine is only fair and the proportion of true oil is small compared with the standard species—the longleaf southern pine. In the Southern States the yield of pure pine oil is 2 to 3.5 U.S. gallons per cord of about 4000 lb. from the best grades

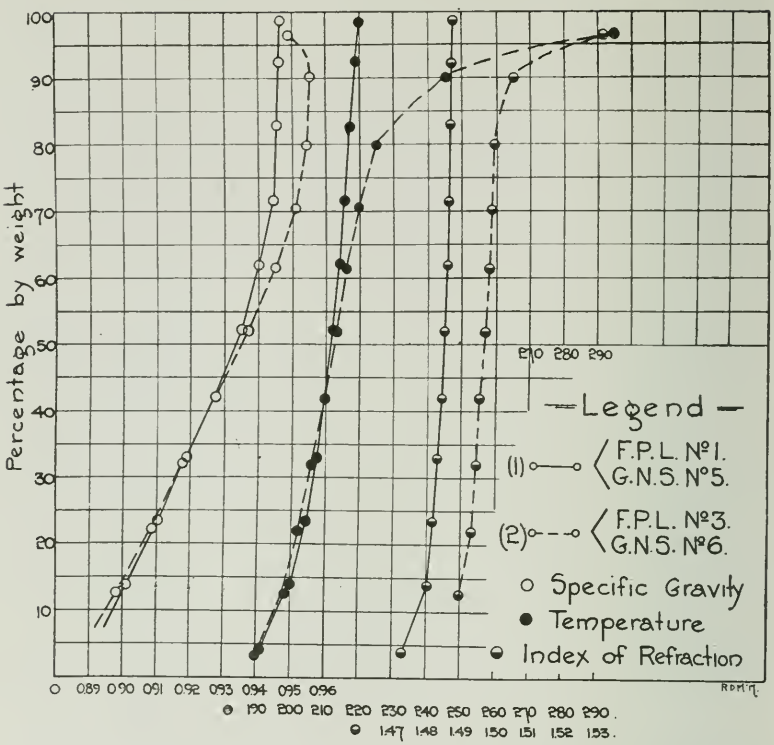


Fig. 1.—Curves showing records of fractional distillation of two different samples of pine oil by steam and solvent process, F.P.L. No. 1 (G.N.S. No. 5); and (2) pine oil by destructive distillation F.P.L. No. 3 (G.N.S. No. 6).

of resinous wood waste. From the above Canadian woods the crude pine oil fraction, containing more or less light oil distilling between turpentine and pine oil proper, amounts to about 4 Imperial gallons per 2600-lb. cord, but only a small proportion of this can be considered as true pine oil. By using two or three times the usual quantity of the sample, fairly satisfactory flotation results were obtained. All attempts to utilize the whole crude oil distillate or its turpentine or resin oil content met with failure in this investigation. As pine oil is essentially a by-product of turpentine and resin recovery from resinous wood waste, and as this recovery industry has not yet shown prospect of profitable establishment in Canada, there is little promise of obtaining a commercial supply of frothing oil from Canadian resinous wood

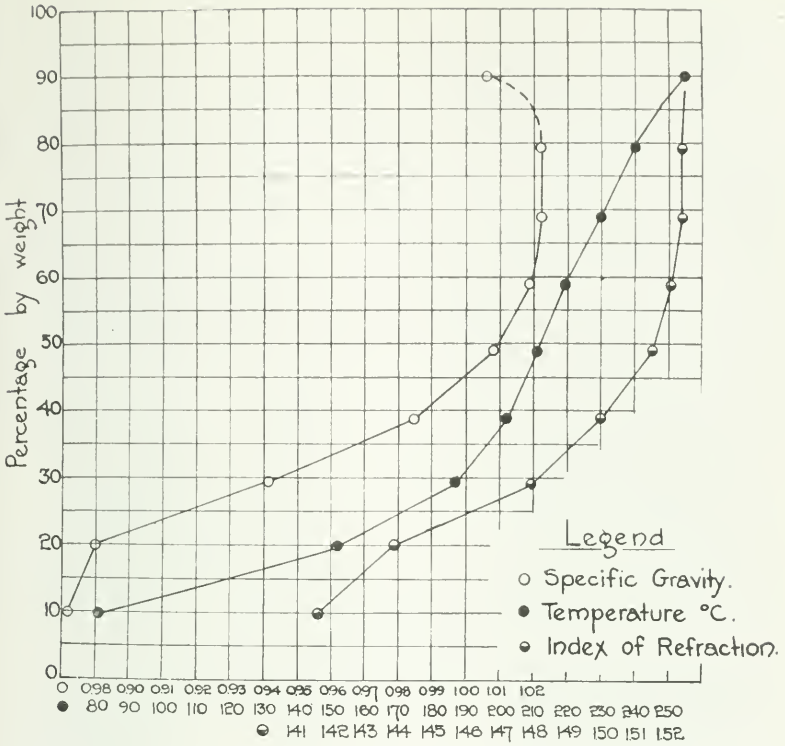


Fig. 2.—Curves showing distillation record of F.P.L. No. 31 (crude hardwood creosote mixture) of which all its fraction except 1 and 2 (i.e. below 20% by weight) gave satisfactory results as a frothing oil in place of pine oil on ore from Cobalt area.

waste suitable for use in the flotation of the ores of the Cobalt area. In this connection the recent statement[1] of Professor H. K. Benson of the University of Washington, who has investigated wood waste industries for the United States Government, is significant, viz.: "The commercial status of the softwood distillation industry is unsatisfactory in the Pacific States, in the Michigan and in the Wisconsin districts." However, considerable commercial progress has recently been made in the Southern States on richer material after a series of failures, and it is to be hoped that technical investigation will in time place the whole resinous wood industry on a sound commercial basis.

[1] U.S. Dept. of Commerce, Bureau of Foreign and Domestic Commerce, Special Agents Series No. 110, "By-products of the Lumber Industry."

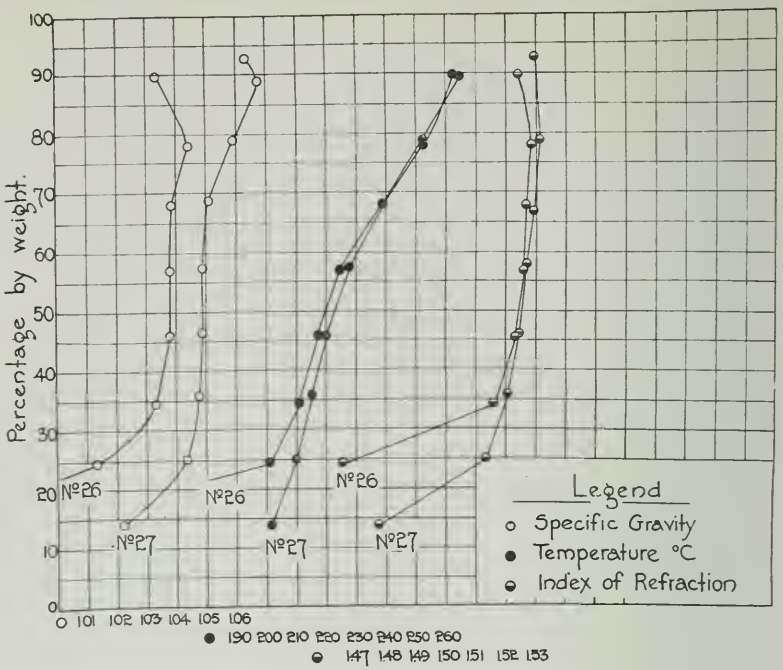


Fig. 3.—Curves showing distillation of crude oils F.P.L. Nos. 26 and 27, heavy hardwood creosote and hardwood 'acid oil' respectively.

Hardwood Oils as Substitutes for Pine Oil.—In the flow sheet (No. 1) for destructive distillation of hardwoods, the creosote oils have been fully outlined as to their source. These hardwood creosote oils, for which there is at present little market value, can be drawn off separately as crude oils in the process of refining the wood alcohol and acetic acid.

The combined production of these three oils, namely F.P.L. Nos. 25, 26, and 27 (combined as oil No. 31), is approximately 2.4 Imperial gallons per 3700-lb. cord and indications are that all three can be used as frothing oils in place of pine oil.

While this yield per cord is about the same as for pine oil from longleaf southern pine, the supply is comparatively large, at present there being over 500 cords of hardwood distilled daily in Canada, and over ten times this amount in the United States, mostly in the border states.

Of the individual hardwood creosote oils heavy hardwood creosote oil No. 26 appeared to give the best laboratory results,

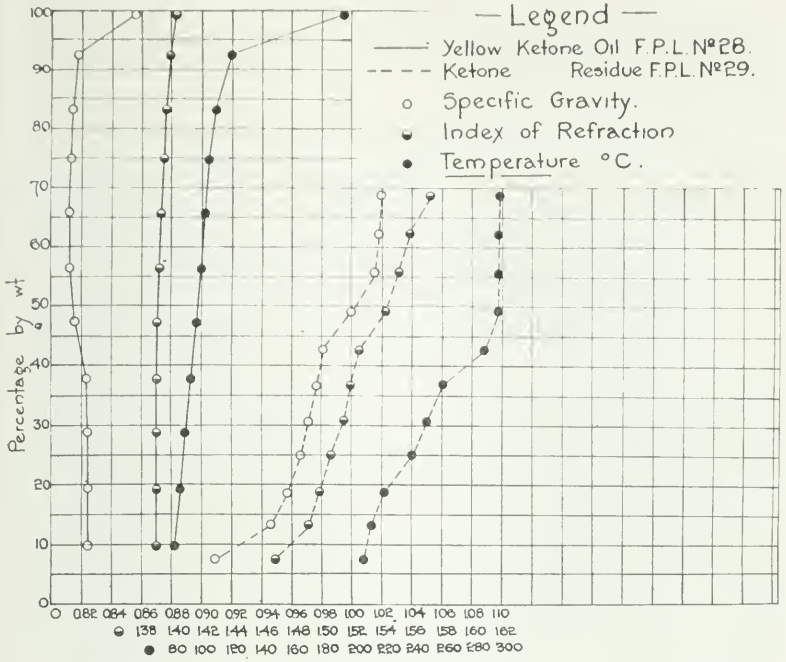


Fig. 4.—Curve showing distillation records of yellow ketone oil F.P.L. No. 28, and crude ketone residue F.P.L. No. 29, both produced in the hardwood distillation refinery.

with acid oil No. 27 nearly as good. Light hardwood creosote oil No. 25, with its lighter fraction boiling below 150°C having gravity below 0.882 removed, gave results comparing favourably with Nos. 26 and 27. This lighter fraction has a disagreeable and penetrating odor and its removal would no doubt make both oils Nos. 25 and 31 (the former oil comprising over 50% of the latter) more agreeable to work with. (See Figures 2 and 3).

Other hardwood oils worthy of mention are heavy yellow ketone oil and ketone residue, by-products of the manufacture of acetone from grey acetate of lime. While the yellow ketone oil is already of considerable value in the industries, its value as a flotation frothing oil is important especially in an alkaline pulp where it gives a high-grade concentrate. Ketone residue, which is black and syrupy in nature, is produced in fairly large quantities and has noticeable collector values, as well as being a good frothing agent. This oil deserves more attention, not only

for use in the treatment of Cobalt area ores, but for use by itself on other ores. (See Figure 4).

Part 2.

Flotation Tests on Silver Ore from Cobalt Area

In all 185 different tests were made, a summary of which is given in Table No. 2. The slime from the Coniagas mill, Cobalt, Ontario, chosen as a representative product of the area, was used in these tests. This slime carried 5.7 oz. of silver per ton, and 62% of the silver was contained in the 200-mesh product. 500 grams of slime, ground to pass 100-mesh, were used in each test. A pulp of one part of water to one of slime along with the carefully measured quantity of oil was ground in a pebble mill for 20 minutes, after which the ground pulp was agitated in the Janney experimental flotation machine for 6 minutes. The temperature during each test varied between 5°C and 15°C.

The Janney testing machine used throughout this experimental work consists of a cylindrical agitation chamber fitted with baffles and a spitzkasten or settling chamber in which the froth collects. The agitation is obtained by an impeller running at 1800 revolutions per minute. A complete description of this machine can be found in the January 1st, 1916, number of the *Mining and Scientific Press.*

It has been recognized, especially for the silver-bearing ore of the Cobalt area, that both a frothing oil and a collecting oil (or oils) were necessary. The oil mixture[1] already giving successful commercial results at Cobalt was as follows:

10% pine oil (pure) as selective frothing oil.
80% coal tar creosote
10% coal tar
} as selective collecting oils.

[1] According to information lately received the majority of the mills have increased the percentage of pine oil to 20% of the total mentioned and are obtaining much better results.

Using this as standard oil mixture No. 1 the procedure was as follows:

1. The substitution for pine oil in this standard oil mixture was tried to a certain frothing value of any new oil. If 10% was not satisfactory the percentage was raised. This constituted the first series of tests on each of the several and different oil samples.

2. Substitutes for coal tar creosote and coal tar in standard oil mixture was tried to ascertain collecting value of any promising oil.

3. Successful frothing substitutes and successful collector substitutes were tried together to ascertain if it was possible to obtain an oil mixture from among the wood oils having both frothing and collecting properties.

The Results.—The aim throughout these experiments was to produce comparative results only, using pine oil and coal tar products as frother and collector respectively as a standard oil mixture. In this connection the reader must recall that no special effort was made to produce high-grade concentrates or to obtain high extractions. A rougher tailing and a rougher concentrate only were produced. Commercial tests on the more promising hardwood oils have already been made, the results of which are presented in Part 3 of this paper.

The results of individual tests varied a great deal, and extractions as high as 80% were averaged with tests giving as low as 60% extraction. To the mining man the efficiency of an oil or oil mixture will be more apparent by comparing the amount of froth floated off and the minimum assay of the tailings.

The more striking features of the results summarized in the table (See pages 78 to 86) are as follows:

1. 67.5% was the average extraction of 20 different tests using pure pine oil (G.N.S. No. 5), where the tails averaged 1.90 oz. from 5.7 oz. heads.

2. Tails as low as 1.14 oz. were obtained by the use of 4 lb. of NaOH per ton of dry ore under the same conditions as above.

3. Pine oil (G.N.S. No. 6) by destructive distillation gave satisfactory results on raising its percentage to 15%–20% of the total oil mixture with over 12% of the coal tar present.

4. The crude oil distillate from the destructive distillation of both western yellow pine and red pine stumps used alone was quite unsatisfactory giving 4.00 oz. tails.

5. Crude turps No. 17 from western yellow pine and a corresponding light oil No. 32 from red pine both had too brittle a froth and left 2.7 oz. tails.

6. Pine tar oil No. 19 from western yellow pine and corresponding oils Nos. 33 and 36 from red pine could not be utilized as frothing oils since they lifted too much gangue.

7. Crude pine oils Nos. 44 and 45 from western yellow pine stumps and No. 35 from red pine stumps gave satisfactory results when they comprised 20%–25% of the total oil mixture. Crude light oil No. 18 from western yellow pine gave comparable results.

8. Results using the three different hardwood creosote oils, namely Nos. 25, 26 and 27 are good. Extractions from 65% to 75% were obtained with tails as low as 1.59 oz. Of the three heavy oils heavy hardwood creosote No. 26 gave the best all-round results. Each of the individual creosote oils gave better results than No. 31, the combination of all three.

9. Oils Nos. 25A and 39 are very similar and deserve attention. They are the hardwood oil fractions distilling below 150°C and having gravity below 0.89. This fraction has a disagreeable penetrating odor and its removal greatly improves oils Nos. 25 and 31 in this respect. The flotation value of this lighter and low boiling point fraction is low, giving 3.9 oz. tails.

10. In general the hardwood creosote oils gave better results in the crude state than when refined by primary distillation.

11. Of the oils Nos. 39 to 43 inclusive, which are fractions distilled from No. 31 (hardwood creosote mixture), all except the lowest boiling fraction No. 39 just mentioned gave approximately the same results.

12. Of the neutral hardwood oil No. 47 (see description) the fraction No. 47C distilling above 200°C gave good standard

results, while the fractions Nos. 47A and 47B distilling below 200°C were non-selective, as shown by 3.0 oz. tailings.

13. Oils No. 48 and 48A being the phenolic and creosotetic content of the crude hardwood oils and practically all distilling above 200°C, gave approximately the same satisfactory results as neutral oil No. 47C having the same boiling point range.

14. Yellow ketone oil No. 28 used alone gave very poor results, but the greatly improved results obtained in a pulp alkaline with NaOH were remarkable, where the extraction averaged nearly 80%.

15. Ketone residue No. 29 which is neutral in character, when used to the extent of 20%–30% of the total oil mixture, gave results comparing favourably with both destructively distilled pine oil and the hardwood creosote oils.

16. Increasing the percentage of coal tar up to 25%–30% of the total mixture invariably gave improved results.

17. By the use of crude hardwood creosote oils the proportion of coal tar creosote could be reduced to 60% of the total oil mixture and still give satisfactory results, but all attempts to eliminate coal tar creosote as the selective collecting oil proved unsatisfactory.

18. Best results were obtained with a cold pulp and at the beginning of the series of tests. The Janney flotation machine gradually decreased in efficiency under continual operation for over five months. This fact is all the more reason for considering the above results as comparative only.

Conclusions.—1. Laboratory results show that crude hardwood creosote oils can be substituted satisfactorily for pine oil as a suitable frothing agent for Cobalt area silver ore.

2. Crude ketone residue from the refining of grey acetate of lime is also a satisfactory frothing oil for use with this ore; heavy yellow ketone oil gives highly satisfactory results when used in a pulp alkaline with caustic soda.

3. The use of NaOH (not more than 4 lb. per ton of dry

ore) gave an almost incredible improvement in extraction and grade of concentrate in all oil mixtures tried.

4. Pine oil from the destructive distillation of Canadian resinous wood waste (western yellow pine and red pine stumps) gave results comparing fairly well with pine oil from longleaf southern pine. Double the quantity, however, of the Canadian product was required.

5. All attempts to utilize the whole crude oil distillates from the destructive distillation of Canadian resinous woods met with failure, as they were non-selective in action. This was also found true of the turpentine and rosin oil fractions of the crude oil.

6. The use of coal tar creosote was found indispensable as a selective collecting oil and all attempts to substitute hardwood tar were unsuccessful. By the use of hardwood oils, however, the proportion of coal tar creosote could be reduced considerably.

Part 3. Commercial Flotation Tests on Hardwood Oils[1]

According to the original plan the most promising wood oils were tested at Cobalt under commercial conditions. Three hardwood oils were given a thorough trial at the Buffalo Mines in comparison with a standard pure pine oil (G.N.S. No. 5). They are as follows:—

F.P.L. No. 26. Heavy hardwood creosote oil.
F.P.L. No. 27. Acid (creosote) oil.
F.P.L. No. 29. Ketone residue.

These oils, which have been fully described in Table No. 1 and Figures Nos. 3 and 4, are quite uniform in the crude state as produced and can be considered standard flotation oils.

Removal of the Aqueous Fraction Advisable.—Samples of the three oils mentioned above when examined at the Forest Products

[1] The hardwood oils used in these commercial tests were supplied from the Longford, (Ontario), plant of the Standard Chemical, Iron and Lumber Company of Canada, Limited (head office, Royal Bank Building, Toronto, Ont.)

Laboratories showed 8.55%, 5.25% and 2.55% water content respectively. The aqueous fractions of the first two oils were distinctly acid in nature and strong enough to have corroding action on ordinary iron metal containers. The water content of the ketone residue was natural. Although these hardwood oils can be used with the small percentage of water present mostly as an emulsion, the removal of the aqueous fraction seems strongly advisable, not only on account of the above objections but by reason of the fact that the acetic acid and its homologues forming the acid content are essentially of no flotation value.

The higher aqueous content of heavy hardwood creosote oil No. 26 compared with acid oil No. 27 is due to its lighter specific gravity. As has been suggested the mixing of these two oils in the proportions in which they are produced commercially would be advisable, since their flotation values are equally satisfactory. In this way a better gravity separation of the water content could be effected than would be possible with oil No. 26 alone.

The yields of these two oils Nos. 26 and 27 as already given are respectively 0.486 and 0.284 Imperial gallons per cord. Figuring on 500 cords of hardwood distilled per day, this means that 385 Imperial gallons of this Canadian wood oil product are available. For about 2000 tons of silver ore treated daily at present, only slightly more than 25% of this oil supply can be consumed in the Cobalt area. Should the demand for these hardwood creosote oils become greater than the supply, it is interesting to note that oil No. 25B according to laboratory results is equally satisfactory. As yet hardwood creosote oil mixture No. 31, comprised of oils Nos. 25, 26 and 27 in proportion produced, has not been tried commercially at Cobalt. Although small scale laboratory experiments on this creosote oil mixture in the crude state show good comparative results, indications are that after the removal of that fraction distilling below 150°C with gravity below 0.89 the mixed residue corresponding to oils Nos. 25B, 26 and 27 would give excellent results with Cobalt area ore. Treating oil No. 31 in this way by a preliminary distillation would also remove all the aqueous content and give a very desirable water-free flotation oil. The commercial supply of this crude hardwood

creosote mixture after such treatment would be over 500 Imperial gallons per day in Canada.

Proposed Specifications for Hardwood Creosote Oils.—Oils Nos. 26 and 27 with 17.5% and 10% respectively distilling below 150°C including 8.55% and 5.25% aqueous content have been found to give satisfactory flotation results when used in the crude state. Calculating to the water-free oil basis this means that these two crude oils contained respectively 9.8% and 4.9% of moisture-free oil distilling below 150°C, according to the standard method of fractional distillation for the examination of crude oils already proposed.

Taking these two crude oils somewhat as a standard and keeping in mind the hardwood creosote oil mixture No. 31 with the aqueous and lighter oil fractions removed as suggested above, it would seem that the following general specifications would be fair restrictions to propose for the buying of hardwood creosote oils, for the purpose of using as selective frothing agents for Cobalt area ore.

1st. The aqueous content shall be less than 5% of the crude oil as received.

2nd. Not more than 10% of the water-free oil shall distill below 150°C (corrected temperature) and with specific gravity below 0.89 at 15°C according to the standard method of fractional distillation already outlined.

3rd. At least 65% of the water-free oil shall distill between 200°C and 265°C, where the gravity shall be between 0.975 and 1.065.

4th. Not more than 15% of the water-free crude oil shall remain as a hard pitch or coke residue.

These specifications would not apply to ketone residue which is somewhat different in nature from the creosote oils. Where a high-grade concentrate is not desired and the aim would be to lift considerably more mineral bearing gangue particles in order to reduce the tailings assay, considerably more pitch residue than

indicated in the fourth specification above may be used. This, however, may be accomplished by adding hardwood tar as such, containing over 70% pitch residue, to the total oil mixture.

Commercial Tests.—The commercial tests, carried on during 42 continuous shifts, were made in the large Callow cells without interruption of the general procedure. During the first 26 shifts the standard pine oil-coal tar products mixture was used. In Tables Nos. 3 and 4 the results are given in full. In the remaining 16 shifts the hardwood oils were substituted for pine oil in order as follows: Heavy hardwood creosote oil No. 26 in 6 shifts, acid oil No, 27 in 5 shifts and ketone residue No. 29 in the last 5 shifts. Tables No. 5, 6 and 7 give the respective results when using these three different hardwood oils.

The tables speak for themselves. It is interesting to note that these commercial tests gave more than 20% higher extractions than were obtained in small scale laboratory tests. This should be borne in mind when studying the comparative results on the fifty different wood oils examined during this investigation.

The operation of the commercial flotation machines needed no special attention during the runs on the hardwood creosote oils Nos. 26 and 27.

When using ketone residue, however, care had to be taken to keep out traces of foreign oil such as lubricating oil which had a deleterious action on the somewhat sensitive though heavy froth.

Conclusions.—1. Crude hardwood oils, the supply of which is abundant, can be substituted for pine oil as a selective frothing agent in the commercial flotation of Cobalt area silver ores.

2. Heavy hardwood creosote oil F.P.L. No. 26 and acid (creosote) oil F.P.L. No. 27 give equally satisfactory results and a mixture of these two oils in proportions as produced is recommended as a standard hardwood flotation oil.

3. Ketone residue in the crude state can also be considered a satisfactory flotation oil for commercial use on Cobalt area silver ore.

TABLE No. 2

[1] The numbers given for the frothing oils in this column are the serial numbers adopted at the Forest Products Laboratories.

Oil Mixture				Per Ton of dry ore		Grams of froth	Assay oz. of silver per ton		Average extraction and remarks
Frothing oil	Per Cent	Collecting oil	Per Cent	Pounds of oil	Reagents		Concentrates	Tails	
No. 1 Pine oil; steam distilled (G.N.S. No. 5).	10	Coal tar Coal tar creosote	10 80	1.5	..	32.0	57.3	1.90	67.5% average of 20 tests
No. 1 Pine oil; same as above	10	Coal tar Coal tar creosote	10 80	1.5	4 lb. NaOH	21.0	105.0	1.14	80.0%
No. 3 Pine oil; Destructive distillation (G.N.S. No. 6)	10	Coal tar Coal tar creosote	10 80	1.5	..	29.0	42.3	3.16	45.0%
No. 3 Pine oil; same as above	15	Coal tar Coal tar creosote	13 72	1.5	..	31.0	61.1	2.04	67.0%
No. 3 Pine oil; same as above	20	Coal tar Coal tar creosote	12 68	1.75	..	49.0	41.7	1.70	72.0% One test only

No. 4 Crude oil from southern pine. (P.T.T. No. 350).	25	Coal tar Coal tar creosote	7.5 67.5	1.5	..	45.0	29.4	2.64	52.5%
No. 4 Crude oil; same as above	100	.....	0	1.5	..	65.0	15.2	4.00	36.0%
No. 14 Crude oil from red pine stumps.	100	.....	0	1.5	..	66.0	17.9	4.00	40.5%
No. 17 Crude turps; western yellow pine stumps.	25	Coal tar Coal tar creosote	15 60	1.5	..	23.0	57.0	2.72	50.0%
No. 18 Crude light oil; western yellow pine stumps.	25	Coal tar Coal tar creosote	15 60	2.0	..	35.0	55.6	1.86	69.0%
No. 19 Pine tar oil; western yellow pine stumps.	10	Coal tar Coal tar creosote	18 72	1.5	..	40.0	33.0	2.66	50.0%
No. 25 Light hardwood creosote	30	Coal tar Coal tar creosote	7 63	1.5	..	21.0	77.0	1.64	67.5%
No. 25A Light fraction of No. 25.	40	Coal tar Coal tar creosote	6 54	1.5	..	8.0	79.0	3.90	25.0% very little froth

Oil Mixture				Per Ton of dry ore		Grams of froth	Assay oz. of silver per ton		Average extraction and remarks
Frothing oil	Per Cent	Collecting oil	Per Cent	Pounds of oil	Reagents		Concentrates	Tails	
No. 25B Heavier fraction of No. 25.	30	Coal tar Coal tar creosote	7 63	1.5	..	32.0	59.0	2.10	66.0%
No. 26 Heavy hardwood creosote.	40	Coal tar Coal tar creosote	6 54	1.5	..	29.5	63.3	2.13	65.0% average of 10 tests
No. 26 Same as above.	40	Coal tar Coal tar creosote	30 30	1.75	..	30.0	70.3	1.54	72.0%
No. 26 Same as above.	30	Coal tar Coal tar creosote	7 63	1.5	4 lb. NaOH	7.0	222.5	1.80	63.0%
No. 26 Same as above.	40	Coal tar	60	1.5	..	28.0	48.0	2.70	52.0%

No. 26A Heavy hardwood creosote distilled.	30	Coal tar Coal tar creosote	15 55	1.5	..	22.5	79.3	2.20	63.0%
No. 27 'Acid oil' (or hardwood creosote.)	40	Coal tar Coal tar creosote	6 54	1.5	..	15.0	139.0	1.8	70.0%
No. 28 Heavy ketone oil.	40	Coal tar Coal tar creosote	6 54	1.5	..	8.0	173.7	3.02	50.0% very little froth
No. 28; Same as above.	26	Coal tar Coal tar creosote	7.4 66.6	2.0	4 lb. NaOH	10.0	235.0	1.30	79.0% average of 6 tests
No. 29 Ketone residue. No. 30 Hardwood tar.	15 15	Coal tar creosote	70	1.5	..	....	42.1	1.96	69.5% surplus froth
No. 29 Ketone residue.	20	Coal tar Coal tar creosote	8 72	1.5	..	21.0	84.7	2.04	64.5%

Oil Mixture				Per Ton of dry ore		Grams of froth	Assay oz. of silver per ton		Average extraction and remarks
Frothing[1] oil	Per Cent	Collecting oil	Per Cent	Pounds of oil	Reagents		Concentrates	Tails	
No. 29 Ketone residue.	30	Coal tar Coal tar creosote	7 63	1.5	..	38.0	55.0	1.64	72.0%
No. 30A Hardwood tar distillate. No. 31 Hardwood creosote mixture.	50 50	.....	0	1.0	..	52.0	25.4	2.72	52.0%
No. 31 Hardwood creosote mixture.	30	Coal tar Coal tar creosote	70 63	1.5	..	20.0	87.7	2.34	61.0%
No. 31 Same as above. No. 29 Ketone residue.	25 25	Coal tar creosote	50	1.5	..	35.0	56.7	2.00	70.5%
No. 31A Hardwood creosote mixture distilled.	30	Coal tar Coal tar creosote	20 50	1.5	..	15.5	110.6	2.54	57.3%

No. 39 First fraction of 31A (28% acidic).	40	Coal tar Coal tar creosote	10 50	1.5	..	6.5	192.4	3.32	43.0% very little froth
No. 40 Second fraction of 31A (33.0% acidic).	40	Coal tar Coal tar creosote	10 50	1.5	..	22.0	83.3	2.00	66.0%
No. 41 Third fraction of 31A (66% acidic).	40	Coal tar Coal tar creosote	10 50	1.5	..	37.0	45.3	2.44	60.0%
No. 42 Fourth fraction of 31A (69% acidic).	40	Coal tar Coal tar creosote	10 50	1.5	..	44.0	42.5	2.12	66.0%
No. 43 Fifth fraction of 31A (50% of acidic).	40	Coal tar Coal tar creosote	10 50	1.5	..	40.0	37.4	2.60	55.6%
No. 32 Light oils from red pine stumps.	10	Coal tar Coal tar creosote	10 80	2.0	..	6.0	232.3	2.76	50.5%

Oil Mixture				Per Ton of dry ore		Grams of froth	Assay oz. of silver per ton		Average extraction and remarks
Frothing oil	Per Cent	Collecting oil	Per Cent	Pounds of oil	Reagents		Concentrates	Tails	
No. 33 Rosin oils from red pine stumps.	10	Coal tar Coal tar creosote	10 80	1.5	..	....	....	4.46	Scum only.
No. 34 Wood turpentine from red pine stumps.		Coal tar Coal tar creosote	10 80	1.5	..	....	....	....	No froth
No. 35 Crude pine oil from red pine stumps.	20	Coal tar Coal tar creosote	8 72	1.5	..	32.0	63.3	2.00	68.5% average of several tests
No. 35 Same as above.	10	Coal tar Coal tar creosote	10 80	1.5	4 lb. NaOH	9.0	235.0	1.52	74.0%
No. 36 Light rosin oil from red pine stumps.	30	Coal tar Coal tar creosote	8 72	1.5	..	31.0	35.0	3.06	42.5%

No. 37 Light hardwood creosote (G.N.S. No. 17).	30	Coal tar Coal tar creosote	7 63	1.5	..	31.0	51.0	2.76	55.0%
No. 38 Heavy hardwood creosote (G.N.S. No. 18).	40	Coal tar Coal tar creosote	6 54	1.75	..	27.0	65.3	2.00	65.0%
No. 44 Crude pine oil from western yellow pine stumps.	20	Coal tar Coal tar creosote	13 67	1.5	..	32.0	48.9	2.44	58.0%
No. 45 Heavy pine oil from western yellow pine stumps.	25	Coal tar Coal tar creosote	11 64	1.5	..	26.0	73.3	1.76	69.5%
No. 47 Hardwood neutral oil from 31A 65° C–290° C.	37	Coal tar Coal tar creosote	6.5 58.5	1.5	..	23.0	72.6	2.34	60.0%
No. 47A Hardwood neutral oil 65° C–150° C.	35	Coal tar Coal tar creosote	6.5 58.5	1.5	..	15.0	65.1	3.20	38.5%
No. 47B Hardwood neutral oil 150° C–200° C.	35	Coal tar Coal tar creosote	6.5 58.5	1.5	..	16.0	81.2	3.08	46.5%

Oil Mixture				Per Ton of dry ore		Grams of froth	Assay oz. of silver per ton		Average extraction and remarks
Frothing oil	Per Cent	Collecting oil	Per Cent	Pounds of oil	Reagents		Concentrates	Tails	
No. 47C Hardwood neutral oil 200° C–280° C.	35	Coal tar Coal tar creosote	6.5 58.5	1.5	..	13.0	128.4	2.00	63.0%
No. 48 Hardwood acidic oil from 31A.	35	Coal tar Coal tar creosote	6.5 58.5	1.5	..	42.0	46.0	2.00	67.5%
No. 48A Acidic oil from 26A and 27A.	35	Coal tar Coal tar creosote	6.5 58.5	1.5	..	32.0	56.7	2.00	66.0%
No. 49 Hardwood neutral oil from 26A and 27A.	35	Coal tar Coal tar creosote	6.5 58.5	1.5	..	26.0	65.7	2.40	60.0%

TABLE No. 3

Oil Mixture—Pine Oil (G.N.S. No. 5)—20 per cent.
Coal tar Creosote 60 " "
Coal Tar 20 " "
Lb. of oil mixture per ton of dry ore—0.98

Date 1917	Shift	Tons of ore treated	Assay oz. of silver per ton				Per cent Extraction
			Heads	Tails	Middlings	Concentrates	
April 16.....	7–3	378	9.8	1.2	16.4	488.0	87.7
	3–7		7.0	1.0	22.2	345.5	85.7
April 17.....	7–3	294	9.4	1.2	20.0	480.0	87.2
	3–7		7.6	1.0	13.4	398.0	86.8
April 18.....	7–3	347	9.2	1.0	12.8	500.0	89.1
	3–7		6.9	1.0	12.0	306.0	85.3
April 19.....	7–3	275	9.8	1.0	12.6	440.0	89.8
	3–7		7.4	1.0	14.0	444.0	86.5
April 20.....	7–3	310	10.4	1.3	14.4	571.5	87.5
	3–7		6.8	1.0	10.6	421.5	85.3
April 21.....	7–3	270	9.6	1.2	11.4	586.0	87.5
	3–7		6–4	1.0	8.8	490.0	84.4
April 22.....	3–7	120	7.0	1.0	11.6	402.0	85.7

TABLE No. 4

Oil Mixture—Pine Oil (G.N.S. No. 5)—20 per cent.
Coal Tar Creosote 60 " "
Coal Tar 20 " "
Lb. of oil mixture per ton of dry ore—1.06.

Date 1917	Shift	Tons of ore treated	Assay oz. of silver per ton				Per cent Extraction
			Heads	Tails	Middlings	Concentrates	
April 23.....	7–3	284	6.2	0.9	8.4	381.0	85.5
	3–7		6.8	1.0	20.0	417.5	85.3
April 24.....	7–3	327	9.0	1.0	9.8	392.0	88.9
	3–7		8.6	1.0	10.0	480.0	88.4
April 25.....	7–3	316	8.6	1.0	10.4	421.0	88.4
	3–7		8.8	0.9	12.8	528.0	89.8
April 26.....	7–3	283	8.2	0.9	8.0	355.0	89.0
	3–7		7.8	0.9	10.0	365.0	88.5
April 27.....	7–3	265	7.8	1.0	8.0	481.5	87.2
	3–7		9.0	0.9	14.9	473.1	90.0
April 28.....	7–3	297	8.2	0.9	9.0	465.0	89.0
	3–7		7.8	0.9	11.6	447.0	88.5
April 29.....	3–7	247	9.0	1.0	16.0	462.5	88.9

TABLE No. 5

Oil Mixture—Heavy Hardwood Creosote Oil (F.P.L. No. 26)—40 per cent.
Coal Tar Creosote 50 " "
Coal Tar 10 " "
Lb. of oil mixture per ton of dry ore—1.06.

Date 1917	Shift	Tons of ore treated	Assay oz. of silver per ton				Per cent Extraction
			Heads	Tails	Middlings	Concentrates	
May 1.....	3–7	172	8.2	1.1	16.0	611.0	86.8
May 2.....	7–3 3–7	263	9.4 8.6	1.1 1.0	17.8 10.0	318.0 436.5	88.6 88.6
May 3.....	7–3 3–7	278	8.8 6.8	1.1 1.0	15.4 11.2	265.0 349.0	87.9 85.5
May 4.....	7–3	94	8.9	1.2	18.4	243.5	86.8

TABLE No. 6

Oil Mixture—Hardwood acid (creosote) oil (F.P.L. No. 27)—40 per cent.
Coal tar creosote 50 " "
Coal tar 10 " "
Lb. of ore mixture per ton of dry ore—1.06.

Date 1917	Shift	Tons of ore treated	Assay oz. of silver per ton				Per cent Extraction
			Heads	Tails	Middlings	Concentrates	
May 4.....	3–7	187	12.0	1.1	92.0	900.0	90.8
May 5.....	7–3 3–7	283	9.6 7.0	1.2 1.0	15.2 22.8	320.0 732.5	87.9 85.8
May 6.....	3–7	125	8.8	0.9	10.6	435.0	89.9
May 7.....	7–3	80	8.8	0.8	10.6	344.0	91.1

TABLE No. 7

Oil Mixture—Ketone residue (F.P.L. No. 29)——25 per cent.
Coal tar creosote 65 " "
Coal tar 10 " "
Lb. of oil mixture per ton of dry ore—1.06.

Date 1917	Shift	Tons of ore treated	Assay oz. of silver per ton				Per cent Extraction
			Heads	Tails	Middlings	Concentrates	
May 7.....	3–7	160	7.8	1.2	26.6	173.0	85.0
May 8.....	11–7	212	9.4	1.0	14.0	317.0	89.6
May 9.....	7–3 3–7	277	9.2 8.2	1.1 0.9	13.0 7.4	267.5 290.0	88.4 89.3
May 10.....	7–3	90	8.8	0.9	87.6	382.0	90.0

Hardwood Flotation Oils for Other Canadian Ores.—Hardwood creosote oils have been tested on a laboratory scale with copper ore from Bruce Mines, Ont. From ore assaying 7.9% copper a mixture of 40% heavy hardwood creosote oil F.P.L. No. 26, 54% tar creosote and 6% coal tar gave 93.1% extraction, leaving tails analyzing 0.8% copper. Other hardwood oils should give equally satisfactory results.

Ketone residue has proved satisfactory for the flotation of low grade copper-nickel ore of the Sudbury district and a commercial demand for the oil has already been effected as the result of this investigation.

Hardwood creosote oils, especially light hardwood creosote oil F.P.L. No. 25, has promising qualities for treatment of the lead-zinc ores and copper ores of British Columbia.

Further investigations of the commercial flotation problems of other Canadian ores, especially those from British Columbia, are now being carried out in the Ore Dressing Division of the Mines Branch, which has secured a complete collection of the Canadian crude wood oils and their fractions.

Acknowledgements.—This investigation was made possible by the hearty support of the Honourable Dr. W. J. Roche, Minister of the Interior, to whom the flotation oil situation was presented by the Cobalt mining companies. The authors wish to express their indebtedness to John S. Bates, Ph.D., Superintendent of the Forest Products Laboratories of Canada, for efficiently planning and directing the investigation throughout, and also to Mr. G. C. Mackenzie, Chief of the Division of Ore Dressing and Metallurgy, Mines Branch, for his co-operation. The commercial tests were made at the Buffalo Mines, Cobalt, with the generous co-operation of Mr. T. R. Jones and his staff. The valuable services of Mr. A. A. Cole, President of the Canadian Mining Institute, are also heartily acknowledged.

This paper is published with the permission of the Director of Forestry and the Director of the Mines Branch, Ottawa.

NOTES ON MILLING PRACTICE AT THE McINTYRE MINE, PORCUPINE, ONT.

By A. Dorfman[1]

The ore milled at the McIntyre may be classified under three heads, depending on the predominating character of the gangue. The three classes are:—(1) a quartz ore, (2) a grey schist ore, (3) an ellipsoidal basalt ore. The principal associated minerals are ferrodolomite, amounting to as much as 80% of the grey schist ore, pyrite varying from 5% up to 20%, also, sericite, leucoxene and chlorite. The specific gravity of the ore depends, of course, on the relative proportions of the different classes described above, and therefore varies from 2.65 up to 2.86 or even 2.90. The grey schist crushes more readily than the quartz; that is a given amount of power expended in crushing the schist will produce a finer product than when expended in crushing the quartz.

Examination of the ore under the microscope, shows that the gold occurs:

(a) As free gold in the quartz,the grains ranging from microscopically fine up to fairly coarse particles.

(b) Occasionally, in the from of fine films in unmineralized schist.

(c) Mainly, in association with iron pyrite.

At the present time, about 80% of the ore milled is a heavily mineralized schist, with a small percentage of quartz. Under the microscope the gold is seen either as very small particles on the corners of the pyrite crystals, or as a fine film on the faces of the crystals. These crystals range from minute to quite visible cubes. The richer ore, however, always contains very minute crystals of pyrite and these are generally found in association with the sc st. The study of the physical characteristics of the ore in conjunction with other experimental work which was started during the mine development period and contained up to this writing has resulted in the development of successful methods for treating the ore. The history of the experimental work may be

[1]Superintendent of the McIntyre Mill.

outlined briefly as follows. The first ore mined at or near the surface was mainly quartz and contained free or visible gold. The treatment indicated as desirable for this ore was amalgamation and concentration.

Amalgamation and Concentration:—A 10-stamp mill and concentration plant was erected on the McIntyre property, designed to treat from 50 to 60 tons of ore per 24 hour day. During the first two months, as much as 80% of the contained gold was recovered by amalgamation, and since from 10% to 15% more was recovered by concentration, there was very little of value left in the tailing. The ore was crushed in the stamps to pass a 20 mesh screen and the average screen analysis of the battery discharge was about as follows:

Screen	% by Weight
+ 40	19.5
+ 60	23.5
+100	25.0
+200	18.2
−200	13.8
	100.0

That much of the gold was granular and coarse as shown by the value of the plate amalgam which ranged from $9.00 to $10.00 per ounce. The ratio of concentration during this period was about 32 into 1. However, such favourable results did not last very long. It was found that the ratio of concentration continually decreased and even by sorting out and thus eliminating from the mill feed the heavily mineralized ore, the ratio finally reached 15 to 1. Less and less gold was obtained on the plates, and a large quantity of concentrate which evidently required special treatment, was accumulated. Altogether the amalgamation-concentration plant was operated for fourteen months, and during the last six months the gold saved or lost could be accounted for as follows:

Gold recovered by amalgamation.....	61.2%
Gold in concentrate.....	24.8%
Gold remaining in mill tailing.....	14.0%
	100.0

For several months before the amalgamation-concentration process was discontinued, the tailing value averaged $1.35 per ton, although very careful attention was given to the concentration of the sulphides and the feed rate to the tables was only 10 tons per 24 hours. The mill feed shortly before this plant was shut down contained about 6.30% FeS_2, of which 5.65% was saved in the concentrate. This would indicate that roughly 90% of the sulphides were recovered by tabling. By hand panning and blanket concentration, very little of the 0.65% FeS_2 in the tailing could be isolated and it was noted that the +60 and +80 mesh particles of gangue still contained small crystals of pyrite. The distribution of the gold value in the tailing according to the size of the particles was as follows:

Screen	Value per Ton.
+40	$1.80
60	1.20
100	1.00
200	0.80
−200	1.40

The gold value in the concentrate was approximately $40.00 per ton.

Cyanidation Tests:—The first experiments were made on the table tailing. A number of leaching tests disclosed the following:

Assay value of tailing before leaching..... $1.35 per ton

		Hours	Per Ton
Gold value extracted by cyanidation	1st	24	$0.20
" " " " "	2nd	24	0.50
" " " " "	3rd	24	0.20
" " " " "		72	$0.90
" " remaining in tailing.....			0.45
	Total.....		$1.35

Leaching tests on the table concentrate were even more discouraging although the concentrate was finer in respect to

size than the tailing. A screen test of the concentrate resulted as follows:

Grade	% by Weight
+30	0.5
40	3.1
60	13.0
100	16.0
100	34.0
—200	33.4
	100.0

Leaching with different strengths of solution up to 10 lb. KCN per ton, resulted in not more than 30 to 45% extraction even in several days of leaching.

Cyanidation by Agitation:—Samples of $10.00 ore were ground to pass an 80 mesh screen and the several grades were cyanided by agitation. The results are given in Table 1.

TABLE 1

Grade	Time of Agitation	Gold Dissolved	Grade	Time of Agitation	Gold Dissolved
+ 100	8 Hours	35%	+200	8	75%
	16 "	65%		16	80%
	24 "	70%		24	83%
	32 "	80%		30	93%
	40 "	95%		36	95%
	48 "	96%			
+ 150	8 "	60%	—200	8	85%
	16 "	80%		16	90%
	24 "	82%		24	97%
	32 "	85%		40	99.7%
	40 "	94%			
	48 "	96%			

This table shows that fine grinding and agitation are essential to obtain a high gold recovery in a reasonable length of time. In this connection, it may be of interest to remark that many experiments indicate that all of the gold in the McIntyre ore is soluble in a cyanide solution, and that this statement is applicable

to the ores mined at other properties in the neighbourhood of the McIntyre mine. The ease with which McIntye gold dissolves in a cyanide solution, can be estimated by the results obtained in the treatment of 1000 tons of concentrate which were accumulated during the operation of the plant already described. Of the $40.00 gold value per ton of concentrate, $38.00 was recovered by grinding the concentrate in a pebble mill, in contact with a cyanide solution (strength 5 lb. KCN per ton) until all sulphide particles were sufficiently fine to pass a 200 mesh screen. The agitation in the pebble mill only, was therefore sufficient to permit the extraction of nearly 95% of the contained gold once the particles were reduced to the size adopted. It is evident therefore that one of the most important factors to be considered in the treatment of McIntyre and other ores in the district is the reduction of the ore to the finest size commercially profitable. The methods now in use at the McIntyre mill to accomplish this and to secure a high gold recovery are as follows:

Mill Practice:—Briefly, the reduction of the ore is accomplished by means of a coarse breaker, rolls, Hardinge ball mills, and pebble mills working in closed circuits with Dorr classifiers. The coarse crusher reduces the run of mine to 3-inch pieces which are further reduced to a maximum of $1\frac{1}{2}$-inch pieces by the rolls. This product is fed to the ball mills which crush to 10 mesh. The final reduction is made by the pebble mills, the finished product from which contains only about 5% of +100 mesh material and 80% of —200 mesh material. On account of the relatively high specific gravity of the sulphides and the operation of the pebble mill in a closed circuit, it is clear that practically all of the sulphides will be reduced to —200 mesh before they can escape from the circuit.

The tonnage crushed by one grinding unit, (1 ball mill and 1 pebble mill) ranges between 150 and 200 tons per 24 hours depending on the average size of final product desired.

Screen analyses of the several feeds, discharges and final pulps, when crushing at the rate of 150 or 200 tons per 24 hours will be found in Table 2. It will be noted that an increase of 50 tons in the feed rate very appreciably affects the character of the final pulp.

TABLE 2

	Hardinge Feed or Roll Discharge	150 Tons per 24 hr.				200 Tons per 24 hr.			
		Hardinge Discharge	Tube Mill Feed	Tube Mill Discharge	Final Pulp	Hardinge Discharge	Tube Mill Feed	Tube Mill Discharge	Final Pulp
On 1½" Ring	1.8%								
" 1 "	17.0								
" ¾ "	16.7								
" ½ "	16.3								
" ¼ "	17.5					3.3%	7.3		
On 8 Mesh	9.6					9.1	5.2		
" 10 "	3.0	1.44%	3.97			6.8	6.5		
" 20 "	4.7	8.65	9.52			19.7	11.6		
" 40 "	3.9	21.0	25.88			19.2	16.4	5.6	
" 60 "	2.0	9.54	5.63	2.24		6.5	12.9	12.0	
" 80 "	0.5	10.64	26.17	15.40		2.6	4.4	7.1	2.5
" 100 "	1.3	4.04	11.14	15.02	5.01	5.2	18.0	22.1	14.5
" 200 "	3.7	6.80	9.02	5.05	13.06	4.2	11.2	22.1	16.0
Through 200 "	2.0	37.88	8.96	52.10	81.93	23.4	6.5	31.1	67.0

The average cost per ton of ore, for crushing, rolling, elevating, ball mill and tube mill grinding, including classifying, for a period of 8 months, at the rate of 400 tons per 24 hours are given in the following table:

	Labour	Supplies	Power	Machine shop	Total
Crushing and Rolling	$0.0354	$0.0182	$0.0101	$0.0068	$0.0705
Ball Mills	0.0248	0.0697	0.0447	0.0090	0.1482
Classifiers and Tube Mills	0.0177	0.0561	0.0870	0.0067	0.1675
Total	0.0779	0.1440	0.1416	0.0225	0.3862

Throughout the 8 months, the mill was not operated at its full capacity. Had this been the case the labour and power costs would have been reduced by at least eight cents per ton, for the total cost of labour and power is practically a fixed quantity. The ball mill, especially in combination with rolls, has proved to be an excellent crushing machine. Aside from such manifest advantages as compactness, noiselessness in comparison with stamps, simplicity of operation and regulation, high running time, the ball mill not only delivers an appreciable percentage of the −200 mesh material required for successful cyanidation, but it also delivers the crushed material in the form of a pulp containing a very small quantity of water. The metallurgical advantages which thus accrue will be apparent, when it is remembered that it is vitally important to reduce as much of the gangue to −200 mesh as can be done economically and if possible all of the pyrite and that the separation of the finished product is attained by classification. The thick pulp from the ball mill is, therefore, of the greatest assistance in classification since it permits a low ratio of water to solids, roughly 3 to 1 or even less, if necessary to secure such a slow classifier overflow current as to enable a +200 mesh sulphide particle to sink with the coarser grains, and be returned to the mill for further reduction. The high classifier efficiency thus made possible is apparent by studying the percentages of FeS_2 in the several grades of the final pulp. For instance, at a 140 ton crushing rate, the percentages determined were:

Grade	% of total FeS_2
+100	2.4
+200	5.4
+200	92.2
	100.0

The distribution of FeS_2 in grinding at the rate of 175 tons per 24 hours will be found in the following Table 3.

TABLE 3

Distribution of Pyrites During Process of Grinding

	Hardinge Discharge			Tube Mill Feed			Tube Mill Discharge			Final Pulp		
	Size %	% FeS_2	% of Total FeS_2	Size %	% FeS_2	% of Total FeS_2	Size %	% FeS_2	% of Total FeS_2	Size %	% FeS_2	% of Total FeS_2
On 20 Mesh	10.0%	7.0%	6.3%	7.0%	16.0%	6.3%						
" 40 "	23.2	8.5	17.8	12.8	14.0	10.0	2.4%	13.0%	2.0%			
" 60 "	13.0	11.5	13.5	14.3	11.0	8.9	12.1	14.0	11.2			
" 80 "	4.3	12.0	4.6	9.6	11.0	6.0	7.2	13.0	6.1			
" 100 "	9.0	14.0	11.4	21.4	15.0	12.4	18.2	15.0	18.0	7.4%	8.0%	5.5%
" 200 "	6.7	13.0	7.9	20.2	27.0	30.7	20.4	19.0	25.6	11.3	9.0	9.4
Through 200 "	33.8	12.5	38.5	14.7	24.0	25.7	39.7	14.0	37.1	81.3	11.2	85.1
	% of FeS_2 11.30			% of FeS_2 17.73			% of FeS_2 15.10			% of FeS_2 10.70		

To secure a high extraction, at least 90% of the sulphides in the overflow should be finer than 200 mesh in size. In the near future wider classifiers will be installed which by decreasing the present velocity of the classifier overflow will eliminate oversize particles in the finished product. The low moisture content in the grinding circuit pulp, makes it possible to further agitate the final pulp without any intermediate settling, if it is profitable to do so.

The two principal features in the present practice are:

(1) Crushing, grinding and at the came time agitating the ore in a cyanide solution (0.6 to 0.75 lb. Na CN per ton of solution).

(2) Recovery of the bulk of the dissolved gold by primary thickeners, the overflow from which goes to the precipitation department, and recovery of most of the remainder, by continuous counter current decantation.

The supplies used per ton of ore treated are:

Cyanide.....0.35 to 0.45 lb.	Pebbles.....4 to 5 lb.
Lime.....2 to 2½ lb.	Steel balls....0.6 to 0.8 lb.
Zinc dust.....0.40 to 0.42 lb.	

Total cost of milling is approximately $0.88 per ton.

Settling and Decantation:—From 77% to 80% of the dissolved values are recovered in the solution from the primary thickener, after which the pulp is washed by means of 4 counter-current-decantation tanks. The 5 tanks treat the final pulp from 1 crushing unit (1 ball and 1 pebble mill). To ensure efficient washing by the decantation plant, the tanks must be designed for the quantity and character of the pulp to be settled and must be operated very carefully. On account of the complexity of the gangue minerals in the McIntyre ore, experiments were made to determine the settling characteristics of the different classes of rock. The results in Table 4 show how different the settling characteristic of the sericite schist is from that of the quartz.

TABLE 4

Quartz			Grey Schist			Quartz Porphyry		Sericite Schist	
Time of Settling	Per cent Solution Clarified	Per cent Solution in Pulp	Time of Settling	Per cent Solution Clarified	Per cent Solution in Pulp	Per cent Solution Clarified	Per cent Solution in Pulp	Per cent Solution Clarified	Per cent Solution in Pulp
0	0.0	75.0	.0	0.0	75.0	0.0	75.0	0.0	75.0
2	4.2	72.3	5	3.4	72.5	1.9	72.7	1.1	72.9
4	8.4	71.5	10	6.6	71.8	3.8	72.4	2.3	72.7
7	13.8	70.1	15	9.2	71.1	5.8	71.9	4.0	72.3
10	19.0	68.8	20	12.0	70.5	6.9	71.7	5.0	72.1
15	27.7	66.3	25	14.2	70.0	8.6	71.3	6.1	71.9
20	35.4	63.0	35	18.0	69.1	11.5	70.7	8.4	71.4
30	49.2	58.0	45	22.7	67.8	14.6	69.9	10.8	70.8
45	60.0	52.1	55	26.1	66.8	17.0	69.3	12.3	70.5
75	70.0	45.0	65	29.2	65.8	19.2	68.7	14.2	70.0
110	75.8	39.7	85	35.0	63.9	23.0	67.7	17.3	69.2
140	77.3	38.1	105	40.0	62.0	26.5	66.7	20.0	68.5
170	78.4	37.0	125	45.0	60.0	30.0	65.6	22.7	67.8
200	79.0	36.4	155	51.1	57.1	34.6	64.0	26.5	66.7
260	79.6	35.7	185	56.1	54 5	38.8	62.5	30.0	65.0
380	80.0	35.3	215	60.9	51.6	43.0	60.8	33. 1	64.5
			245	64.8	48.9	47.1	59.0	36.9	63.2
			305	71.9	43.2	55.7	54.7	43.8	60.5
			365	74.2	41.3	63.2	50.1	49.6	57.8
			455	75.0		71.7	43.6	60.4	51.9
			485	75.0		71.9		65.2	48.7
			605	75.2		72.5		66.9	47.4
			755	75.4		72.8		67.1	
			905	75.7		72.9		67.3	
			1085	75.7	40.0	73.0	42.4	67.3	47.0

It will be noted that whereas the quartz will settle to a 35% moisture pulp in 3½ hours the grey schist pulp in the same time will contain 51% of water, quartz porphyry 60% and the sericite schist 64% water. Even after 18 hours of settling, sericite schist pulp still contains 47% moisture.

The thickness of the pulp discharged from a tank cannot be regulated at will by the operator beyond a certain limit. If it is attempted to discharge a pulp thicker than the critical or maximum working thickness, and this no doubt is governed by the settling characteristic of any given pulp, it will be found that the solids will soon make their appearance in the overflow. Figure 1 discloses the conditions in a tank, 11 feet deep and 29½ feet diameter, when in good working order, as shown by samples taken 3 feet apart at each foot in depth.

It will be noted that there are practically 4 zones in this tank as follows: First, there is about 5 feet of clear solution at the top of the tank, then about 1 foot of very light pulp containing about 2% colloidal material, then there are 2 or 3 feet of pulp in different stages of settling and finally about a foot of settling pulp over the rakes which contains about 60% solids or 40% moisture. Figure 2 shows a tank in bad order since there is only about a foot of clear solution at the top.

When a thickener is working well, there should be between 3 and 4 feet of clear solution below the overflow, consequently it might be inferred from the data in Figure 1, that the depth of this tank could be decreased by one or two feet without disturbing the settling operation. The expulsion of liquid particles is slower in a deep than in a shallower column of pulp, as can be noted by filling two graduates with identical samples of pulp, and inclining one from its vertical position. It will be observed (see Fig. 3) that a greater quantity of clarified solution has separated out in the inclined than in the upright graduate, the time of settlement being the same for both. The explanation of this is that the depth of the pulp is less in the inclined graduate.

There is some difference of opinion between mill operators regarding the most efficient depth and diameter of thickeners. In support of the result noted in the little experiment just described, it might be mentioned that the first thickeners installed at the

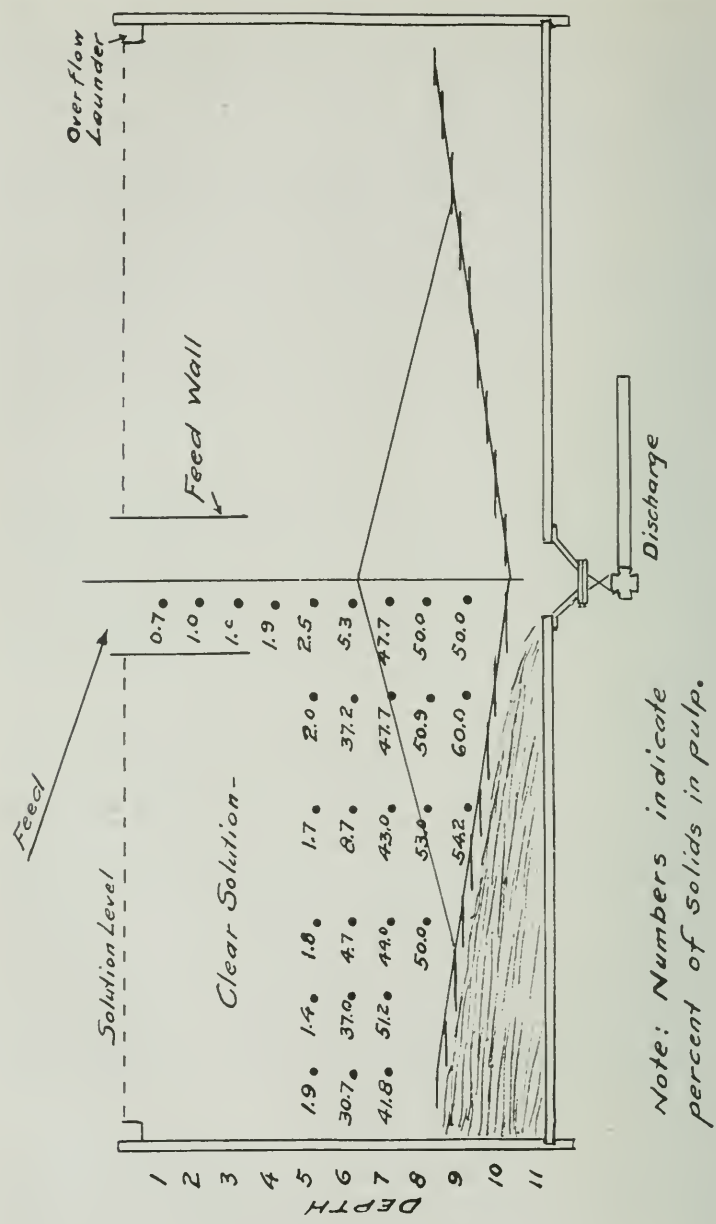

Overflow Launder
Feed Wall
Discharge
Feed
Solution Level
Clear Solution
DEPTH
1 2 3 4 5 6 7 8 9 10 11
0.7 1.0 1.0 1.9 2.5 5.3
1.9 1.4 1.8 1.7 2.0 3.7 4.7 47.7 50.0
30.7 37.0 4.7 8.7 37.2 47.7 50.9 50.0
41.8 51.2 49.0 43.0 47.7 53.0 54.2 60.0 50.0
50.0
Note: Numbers indicate percent of solids in pulp.

FIG. 1

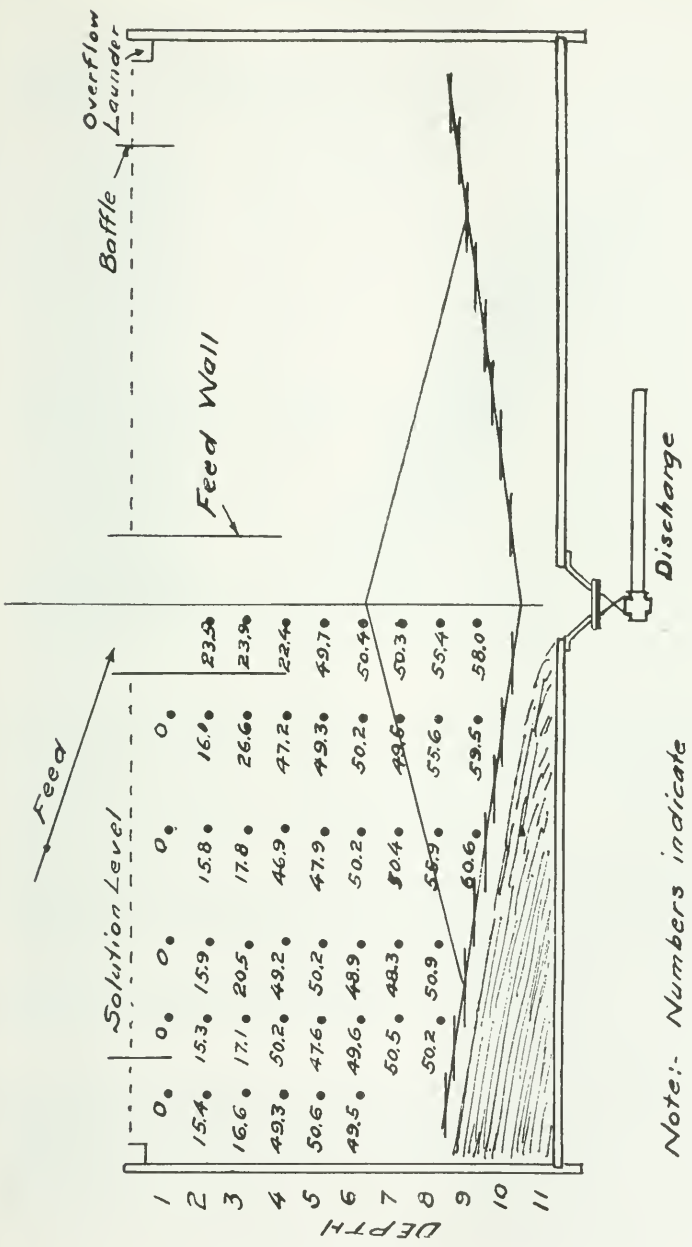

Overflow Launder
Baffle
Feed Well
Discharge
Feed
Solution Level
DEPTH
1
2
3
4
5
6
7
8
9
10
11
0. 0. 0. 0.
15.4 15.3 15.9 15.8 16.1 23.5
16.6 17.1 20.5 17.8 26.6 23.9
49.3 50.2 49.2 46.9 47.2 22.4
50.6 47.6 50.2 47.9 49.3 49.7
49.5 49.6 48.9 50.2 50.2 50.4
50.5 48.3 50.4 49.6 50.3
50.2 50.9 56.9 55.6 55.4
60.6 59.5 58.0
Note:- Numbers indicate percent of solids in pulp.

FIG. 2

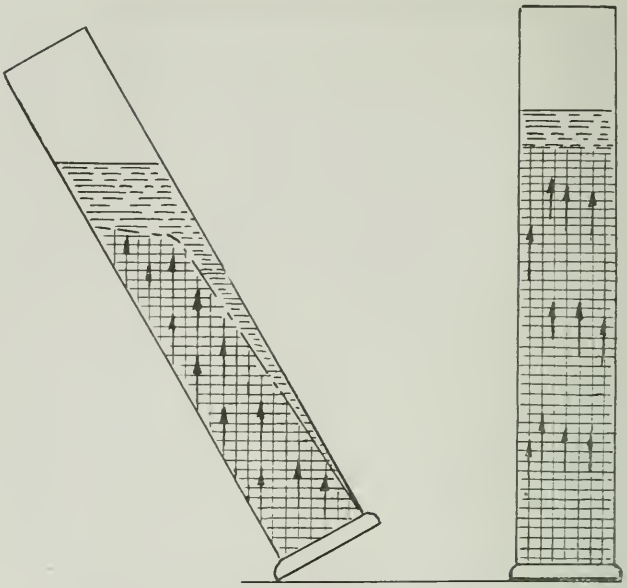


FIG. 3

McIntyre were 11 feet deep. The last two units erected are only $7\frac{1}{2}$ feet deep and they are doing much better work than the deeper ones.

While the pulp zones in a thickener are horizontal, to within 3 or 4 feet of the centre of the tank, similar sized grades are found on lines downwardly inclined from the centre as shown by screen tests of pulp samples taken in different places in a cross-section of a tank. The results are given in Figure 4.

This is due to the diminishing velocity of the current as it approaches the overflow lip.

The relative capacities of thickeners settling McIntyre ore may be guaged by the following data: Thickeners, 32 feet in diameter will settle 175 tons of ore in 500 tons of solution per 24 hours and yield a clear overflow, whereas the tanks, 30 feet in diameter, will only handle 140 to 150 tons of the same ore in the form of a similar pulp.

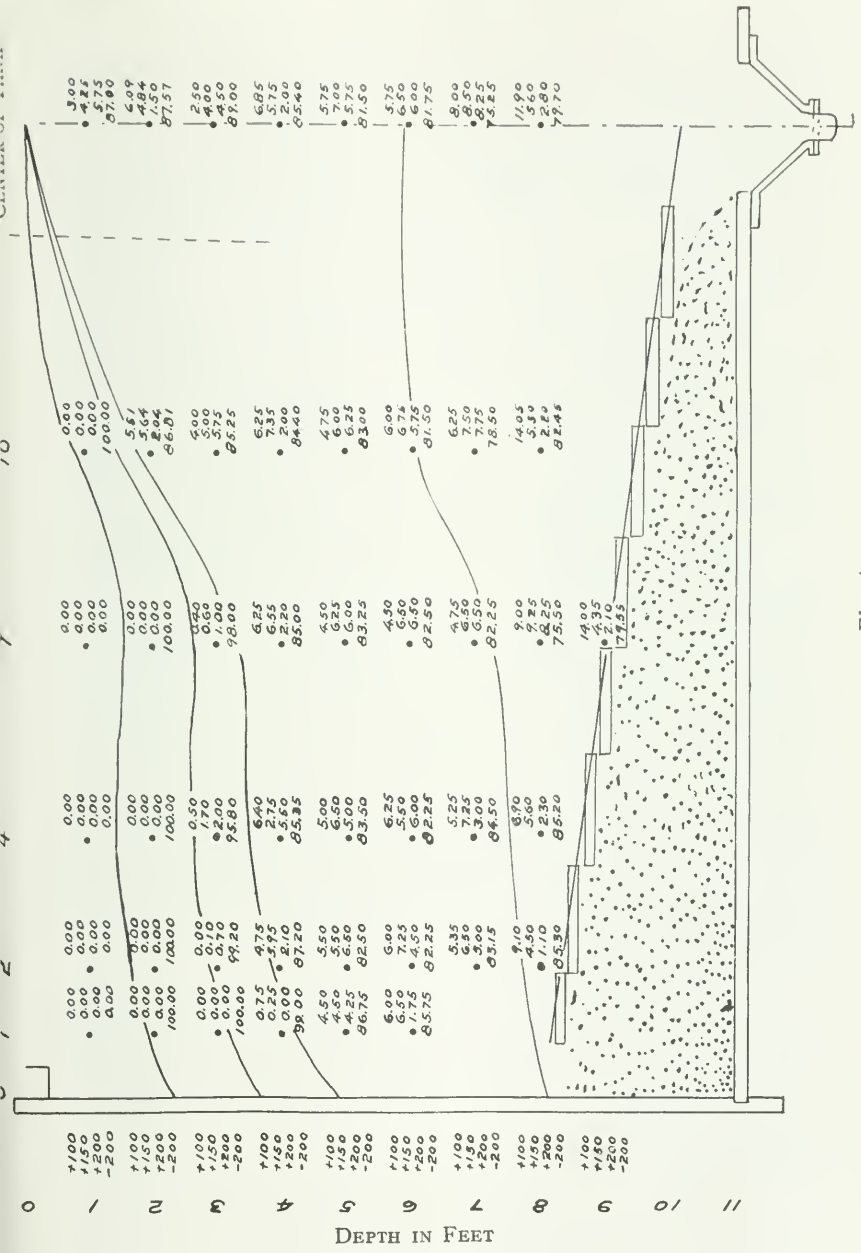


Fig. 4

DISCUSSION

MR. JOHN W. BELL: There is, undoubtedly, a large amount of interesting information concerning the milling practice in the Porcupine district which has not, as yet, appeared in print, and it is to be hoped that other operators in that district will follow Mr. Dorfman's most excellent example and give us the benefit of their valuable experience just as Mr. Dorfman has given us some of his, in the paper under discussion.

Being specially interested in problems in rock crushing, my attention has been particularly directed to Mr. Dorfman's description of the McIntyre crushing plant. It is a departure from the older practice and while as yet, the available data is too limited to warrant the statement that the ball mill is mechanically more efficient than the stamp mill it replaces, I am inclined to think that before long data will be produced to prove definitely that it is. Certainly, operators in the Porcupine district should spare no effort in studying ways and means for reducing the cost of crushing their ores, since, by Mr. Dorfman's figures nearly half of the total cost of milling is represented by this single item. I cannot but believe that much more rapid strides will be made in increasing the efficiency of crushing plants, when operators shall commence to study the performance of crushing machines and their combinations in a way, at once more scientific and definite than has been employed in the past. Now that we can feel confident of the practical accuracy and reliability of Rittinger's theory, there is no longer any reason for neglecting to make use of it and I am confident that if mill engineers will do this, it will be very helpful to them in solving some of the many problems they are constantly confronted with.

Quite recently I had occasion to calculate the efficiency figures for a number of ball and pebble mill tests by this method and perhaps the most striking feature brought to my attention by these calculations, was the relatively great inefficiency of a ball mill crushing a fine feed in comparison with the efficiency of a pebble mill of approximately the same size. Roughly, the pebble mill crushing a 50 mesh feed was mechanically nearly twice as efficient as a ball mill crushing a considerable coarser feed and had the feed to the two machines been of the same size, the comparison

would have been still favourable to the pebble mill. The reason for this is I think quite simple since the pebbles did almost as much crushing as the balls with an expenditure of about half the horse power required to lift the balls. The point, therefore, that I would like to bring to the attention of Mr. Dorfman and other operators in the Porcupine district, is, that while I have no reason to doubt that the ball mill will prove its high efficiency when crushing the coarse feed formerly sent to stamps, I am of the opinion that to attain maximum ball mill efficiency it will be advisable to feed these machines at such a high rate as to produce a much coarser discharge than −10 mesh. By so doing, it is probable that not only will the efficiency of the ball mill be increased but that the efficiency of the secondary pebble mills will also be increased. Practical experience indicate that the efficiency of a pebble mill is greatly affected by the size of the feed delivered to it, that is if a pebble mill can efficiently crush $\frac{1}{4}$-inch pieces of a given ore, its efficiency will be less when crushing 10 mesh material and still less if crushing very small grains. Just what maximum size of ball mill discharge or pebble mill feed, will secure the highest all round efficiency, will, of course depend on the resistance to crushing of a given ore, but it is probable that the −10 mesh feed to the McIntyre pebble mill, could be raised to $\frac{1}{4}$-inch or even a greater diameter. It might therefore be profitable to make some experiments along the lines suggested since by making 1 ball mill crush let us say 300 tons of the rolls product, and by sending the ball mill discharge to 2 secondary pebble mill (150 tons each), it is at least possible that as much or nearly as much grinding might be done as it now accomplished by 2 ball mills and 2 pebble mills. What the outcome would be in an all-sliming plant is by no means certain but it may be worth investigating. It is probable that the size and design of the ball mill would have an important bearing in the results that would be obtained.

NOTES CONCERNING LOCAL TREATMENT FOR CERTAIN WEST KOOTENAY SILVER ORES.

By Francis A. Thomson,

Mining Engineering Dept., State College, Pullman, Wash., U.S.A.

Western Branch, Trail Meeting, Oct. 1916.

The notes herewith are submitted not with any exaggerated idea of their intrinsic value but solely in the hope that by saving them from the oblivion of a dusty file some slight service, even if of a negative kind, may be rendered. Nor is it the desire of the writer to convey the impression that either of the properties mentioned would necessarily justify the erection of a treatment plant along the lines indicated by the experiments, but it is felt that the results here set forth may possibly stimulate owners of mining properties having similar ores to investigate the advisability of local treatment. It is notable that operating cyanide plants are extremely rare in British Columbia, and it is an open question whether such a condition is justified.

Tests on Silver Glance Ore.

The first set of experiments here reported is one made more than six years ago on samples of ore from the Silver Glance mine, above Bear Lake, a station about 20 miles from Kaslo, British Columbia, on the Kaslo & Slocan railway. The ore in this and adjacent properties is what is known as a 'dry' silver ore, that is, one having insufficient lead or copper to collect its own gold or silver should it be smelted with suitable flux.

An analysis of the sample used for the treatment tests showed:

SiO_2	96.1 per cent.
Al_2O_3	2.3 "
Fe_2O_3	1.4 "
Moisture	1.1 "
Total	100.9
Silver	18.4 oz.
Gold	0.02 oz.

From the dendritic markings the presence of manganese was presumed but the amount was too small to appear on general analysis.

The silver occurred mainly as cerargyrite (silver chloride) and argentite (silver sulphide). It was thought that such an ore would prove readily amenable to cyanide treatment, since both cerargyrite and argentite are easily soluble in cyanide solutions.

Results on small scale tests were as noted below:

Head Assay—18.4 oz. Ag.

Comminution	Agitation Time. Hours	Solution Strength. Per cent.	Lime 16 lb. per ton.	Tailing Assay. Oz.	Extraction Per cent.
100	8	0.25	5	14.4	21.8
100	8	0.25	10	15.1	17.9
100	8	0.25	15	12.1	34.2
100	8	0.25	20	12.0	34.8
100	8	0.50	10	11.0	40.3
100	8	0.50	20	10.0	45.7
100	8	0.75	20	10.2	44.5
100	24	0.75	20	9.0	50.5
100	8	1.00	20	9.0	50.0
100	24	1.00	20	7.2	61.0
150	16	0.50	10	7.4	59.8
150	16	0.50	20	5.3	71.2
200	16	0.50	10	7.6	58.7
200	16	0.50	20	6.1	66.9
200	24	0.50	20	5.8	68.5
200	16	0.50[1]	15	4.2	77.2
200	16	0.50[2]	10	2.1	88.5
200	16	0.50[2]	20	1.6	91.5

[1]Lead Acetate added in proportion of 0.35 per cent.
[2]Litharge added in proportion of 0.25 per cent.

Confirmation tests on 10-lb. lots of ore were run on the basis of the preceding with the following results:

Conditions: Head assay, 19.1 oz.: Comminution 91%

through 200 mesh; Cyanide solution, 0.50% KCy; Lime, in proportion of 20 lb. per ton.

Agitation Time. Hours.	Cyanide Consumption. Lb.	Extraction Recovery. Per cent.	Additional reagents. Per cent.
16	1.8	72.4	None
24	2.1	74.1	None
16	1.6	80.9	Lead acetate 0.4% of ore
24	1.4	84.3	Lead acetate 0.4% of ore
16	1.5	89.5	Litharge 0.3% of ore
24	1.6	93.1	" 0.3% of ore

Concentration experiments proved futile as expected, since there were practically no sulphides present except the silver. For the same reason it is not thought that flotation would be applicable to such ores as this. However, it is evident that a satisfactory extraction may be expected by straight cyanide treatment, and without an undue consumption of cyanide.

If such an ore as this were available in sufficient quantity to justify a treatment plant having a capacity of 100 tons a day, I believe that, by careful management, a cost of $2.50 a ton could be attained.

TESTS ON SILVER HOARD ORE.

The sample submitted consisted of 500 lb. of unoxidized ore taken from the 50-ft. crosscut, east of the high-grade shoot. The visible minerals were: Pyrite, (iron bisulphide), blende (zinc sulphide), and galena (lead sulphide) contained in a gangue of calcite, dolomite, and some siliceous schist.

An analysis of the ore resulted as follows:

SiO_2	28.41	per cent.
Al_2O_3	12.24	"
MgO	2.32	"
CaO	12.55	"
Fe	8.74	"
Zn	12.50	"
Pb	0.50	"
S	11.20	"
Ag	25.2	oz. per ton
Au	0.015	oz. per ton

Unaccounted for, carbon dioxide (Co_2) and water.

To determine the locus of the silver in the ore, the various visible minerals were carefully segregated and the segregate products separately assayed. The results were as follows:

General sample	25.2 oz. silver per ton
Segregated blende	10.0 " " " "
Segregated pyrite	46.5 " " " "
Segregated galena	46.5 " " " "
Calcite (showing native silver)	101.4 " " " "

From this it will be seen that quantitatively the pyrite and the calcite are the important sources of silver, since the blende is low in silver and the galena small in amount.

Three methods of local treatment are possible, namely, amalgamation, concentration, and cyanidation, either singly or in combination with each other.

The recovery of silver from its ores by amalgamation is a process rapidly passing into disuse; it was thought best, however, in view of the large amount of native silver in this ore, to see what could be done.

The results are shown below:

Amalgamation of raw ore.

Head assay 25.2 oz. silver.

Method of Treatment	Tailing Assay.	Recovery
Crushing with mercury in stamp battery, followed by apron plates; 40-mesh screen used	15.6 oz.	38.1 per cent.
Previously crushed to 40-mesh and then allowed to flow once over amalgamation plates	25.2 oz.	nil.
Previously crushed to 40-mesh and then allowed to flow three times over amalgamation plates.	24.1 oz.	4.3 per cent.

Previously crushed to 40-mesh and then allowed to flow five times over amalgamation plates.	23.6 oz.	6.3 per cent.
Bottle test, ore crushed to 40-mesh and agitated two hours with mercury.....	23.4 oz.	7.1 per cent.

After a series of small-scale tests to determine the most desirable conditions, a sample of 100 lb. was crushed to 20 mesh in the rolls and used as 'natural feed,' that is, unsized and unclassified, to a 6-ft. Wilfley table. The results were as follows:

Product.	Weight Lb.	Lead Per cent.	Iron Per cent.	Zinc Per cent.	Silver Oz.
Concentrate.....	1.8	17.2	8.3	9.8	161.5
Middling	39.3	0.1	11.7	18.8	40.6
Tailing.....	41.1	0.0	5.6	3.4	9.1
Slime.....	18.8	0.5	7.3	5.3	14.0

To test the adaptability of this ore to cyanidation, samples of 10 lb. each, were prepared and, after preliminary tests to determine solution strength and lime requirement, were tested with the results shown below.

Cyanidation of Raw Ore

Head Assay 25.2 oz. Ag.

Solution 0.5% KCy Lime 5 lb. per ton.

Comminution of Ore.	Hours Agitation.	Cyanide Consumption.	Tailing Assay.	Extraction
100 mesh	24	3.0 lb.	3.5 oz.	86.1 Per cent.
100 "	36	3.0 "	2.9 "	88.5 " "
120 "	24	3.0 "	2.8 "	88.7 " "
120 "	36	3.1 "	2.8 "	88.7 " "
150 "	18	2.8 "	2.3 "	90.8 " "
150 "	24	3.0 "	1.9 "	92.5 " "
200 "	18	2.5 "	1.6 "	93.6 " "
200 "	24	3.4 "	1.2 "	95.2 " "

An approximate screen analysis of each of the products cyanided was as follows:—

Product	−100+120	−120+150	−150+200	−200	
100 mesh	16.6	8.3	8.3	66.6	Per cent.
120 "	....	8.3	16.6	75.0	" "
150 "	....	....	12.5	87.5	" "
200 "	....	....	....	100.0	" "

Cyanidation of Tailing from Amalgamation

Head Assay 15.2 oz. Ag.
Solution 0.5 per cent KCy Lime 5 lb. per ton.
All crushed to pass 150 mesh.

Hours Agitation	Cyanide Consumption	Tailing Assay	Extraction
18	3.1 lb.	1.6 oz.	89 per cent.

Cyanidation of Tailing from Concentration.
Head Assay 9.1 oz. Ag.
Solution 0.5 KCy Lime 5 lb. per ton.
All crushed to pass 150 mesh.

Hours Agitation	Cyanide Consumption	Tailing Assay	Extraction
18	1.2 lb.	0.5 oz.	94.4 per cent.
24	1.8 "	0.3 "	96.7 " "
48	2.0 "	0.1 "	98.9 " "

Deductions from Tests.

From the preceding tests the following deductions may be drawn:

I. *Amalgamation*, while yielding a reasonable recovery under the best conditions, is dependent for its success upon the silver occurring in the native form, and since this is probably of secondary origin, I would not consider its continuance to deep levels as likely; hence I believe that amalgamation may well be omitted from any plans for treatment of this ore.

II. *Concentration*, while perhaps not an essential feature in the treatment of such ore as described, will certainly become necessary in the treatment of the ore from greater depths. That it is feasible is demonstrated by the test reported herewith, the results of which show that with more refined methods a reasonably good recovery by concentration may be expected.[1]

As to the desirability of concentration in preference to direct cyanidation, this will depend upon two main factors—

(1) The changes in the character of the ore as depth is gained; and (2) the condition of the market for zinc concentrate. Summarizing it may be said that while concentration is feasible, it may or may not be desirable.

III. *Cyanidation.* As has proved to be the case in Mexico and Nevada, the silver ores of Ainsworth camp are evidently amenable to cyanidation. Strong solutions 0.5 KCy, (10 lb. to the ton) are probably necessary and the cyanide consumption on raw ore is high; this would probably be less under mill conditions than my laboratory tests would indicate. Reasonably fine grinding from 120 to 150 mesh is apparently necessary.

An interesting feature of the tests is the extremely high recovery yielding by combined concentration and cyanidation, (99.6 per cent) with, it will be noted, a marked decrease in cyanide consumption. The cost of treating such an ore as this would be higher than for a simple ore such as that of the Silver Glance mine.

Probably $3 to $3.50 per ton would be a reasonable cost figure, including concentration but excluding treatment of the concentrates.

Conclusion

In conclusion it is evident that, did conditions warrant, both of the ores here discussed might be treated at the mine instead of being shipped to the smeltery; as to whether such would be advisable is of course another question.

Reference has been made earlier to the rarity of cyanide plants in British Columbia and, in fact one might add, of all

[1]Written more than three years ago. It is possible that flotation might raise concentration recovery to a point rivalling that by cyanidation.

metallurgical plants designed for final extraction, aside from the four or five large smelting works.

So far as the Kootenay districts are concerned, this is doubtless to be explained in part, at least, by the favourable freight and smelting rates granted by the Canadian Pacific Railway and the Consolidated Mining and Smelting Co., and by the encouragement thereby given to the smaller operators to ship their ore rather than to treat it at or near the mine.

Such a condition is vastly to be preferrred to that so common in the western United States where apparently the erection of a local treatment plant was the first thing to be decided upon, leaving for subsequent consideration the questions of suitability of process and adequacy of ore supply. At the same time, as the mineral resources of the province are developed, it will doubtless come to be recognized that the most favourable freight and smelting rates cannot, in the case of ores in which gold and silver are the predominantly valuable constituents, unless these ores are unusually refractory, compete with standard processes such as amalgamation and cyanidation aided by concentration.

Lest I be misunderstood in this matter, however, let me say in conclusion that in no case would I recommend the erection of a local treatment plant until there is positively blocked out in the mine and on the dumps, ore sufficient to keep the contemplated plant in operation for at least one full year, and this would mean a concomitant amount of 'probable' and 'possible' ore in the mine.

NOTES ON THE HIGHLAND VALLEY MINE AND MILL, NEAR ASHCROFT, B.C.

By Frederic Keffer

The mining claims now being developed and worked by the Highland Valley Mining & Development Company are situated about 25 miles southeasterly from the town of Ashcroft. The distance by waggon road is 30 miles. The elevation is 5500 feet above sea level; and, although within the dry belt of British Columbia, at this altitude there is much snow and severe weather during the winter.

The main ore body, so far as explored, lies along the north contact of a coarse-grain granite, containing much biotite (the prevailing country rock of the district) and a broad belt of quartz-feldspar, almost porphyritic in its general appearance. The strike is easterly and the dip about 80° to the south. A branch of the vein extends toward the southeast following a narrow and very much altered dike of basic composition. This dike lies within the quartz-feldspar; and on both sides of the dike are orebodies, the dike itself being fairly well mineralized.

The main economic mineral of the ore is copper, occurring as chalcopyrite; small quantities of gold and silver are also present, particularly in those localities where the ore contains abundant quartz. For the most part, the gangue of the vein is micaceous, in which muscovite predominates, while biotite is also present. Considerable quantities of iron (as hematite) and lesser amounts of quartz and feldspar are also found in the gangue. The mineralization along the contact is chiefly within the quartz-feldspar. The ore is rather remarkable in that it contains practically no metallic sulphides other than chalcopyrite, so that, when milling, the table concentrate contains chalcopyrite and hematite, whilst the flotation concentrate contains little save the copper sulphide and more or less micaceous gangue.

The vein outcrops at intervals for a distance of 660 feet; to date it has been followed underground 400 feet. In this distance several minor faults have been encountered. The vein varies from 3 feet to 17 or 18 feet in width, the average width

along the main adit tunnel being about 7 feet. The copper contents of the vein are about 5% but the ore, as mined and sent to the mill, assays from $3\frac{3}{4}$ to $4\frac{1}{4}$% of metallic copper; the lower value is due to greater or lesser inclusions of wall rock in the milled ore. Surface erosion seems to have stopped at the point where the vein was narrowest, for everywhere the vein has widened with depth. No material difference had been noted between the grade of the ore of the tunnel level and that of the surface. At present all mining is by hand. The ore is conveyed to the mill, 1000 feet distant, by a rope tramway actuated by gravity.

The fact that practically the only metallic sulphide present is chalcopyrite and that this is contained in soft micaceous gangue, has made the milling of the ore a comparatively simple problem. The ore is automatically dumped from the tramway cars into the mill bin, which has a capacity of 300 tons. From this bin it is fed into a Blake crusher, having a 9 ins. x 16 ins. jaw opening, which reduces the ore to $1\frac{1}{4}$ in. size. The ore then passes through a set of 12 ins. x 30 ins. rolls, to an elevator, which delivers into a trommel screen of 3-millimeter mesh, the oversize being returned to the rolls. The undersize from the screen drops into a fine ore bin of 70 tons capacity, from which it is fed to a 6 ft. x 22 ins. Hardinge pebble mill. The oversize from Hardinge is returned to the feed end, by a belt drag classifier. From the Hardinge mill the ore goes directly to a pair of Butchart tables, only one of which, however, is used at a time.

When the mill was first started the slime was removed from the discharge of the Hardinge mill by an Anaconda deslimer, but experience showed this to be entirely unnecessary, as the Butchart tables did as good work on unclassified ore directly from the Hardinge mill as they did when the slime was removed.

The copper mineral extracted on the tables amounts to between 40% and 50% of the total copper contents of the ore, the greater part of the remaining copper being contained in the slime and very fine sands. This is, undoubtedly, due to the fliable condition of the chalcopyrite as it occurs in the ore.

The tailing from the tables goes to a 'Minerals Separation' type of flotation machine, made up of 10 cells each 18 ins. square, the first four of which are provided with spitzkastens. The

table tailings enter No. 5 cell and pass on to No. 10, from which the final tailing pass to the waste dump. The collected froth from these six cells goes to an elevator that discharges into No. 1 cell, and is cleaned in cells 1 to 4, the froth being removed from the spitzkastens by mechanical skimmers.

From the spitzkastens the froth passes into a well, from which it is pumped into a settling tank and, after thickening, is run to an Oliver filter. The product from the filter is dried and sacked for shipment.

The total extraction ranges from 85% to 94%, depending somewhat on rate of feed to the Hardinge mill. With very finely ground ore there is a higher extraction in the flotation machine, but, as very fine grinding involves increased cost and decreased output, there is a point, between finer grinding and coarser grinding with increased output, at which the greatest profit is obtainable, which point seems to be in the neighborhood of 91% or 92% extraction.

Numerous varieties of oils have been tested in the flotation plant, and to the present time the best general mixture has proved to be stove oil, fuel oil and North Carolina tar (Stockholm tar). Stove oil, pine oil and a very little turpentine also make an effective mix. Addition of sulphuric acid does not appear to aid extraction to any noteworthy extent.

Flotation concentrates carry between 20% and 30% copper, the chief impurity in the product is very small scales of mica which have more or less tendency to float along with the copper sulphides.

The entire flotation machine has been in operation only a short time and there is, of course, still much trying and fitting to be done in order to reach a point where a maximum grade of product, together with minimum tailing loss and minimum oil consumption, may be had.

The power used in the mill is steam and the fuel is cordwood. During the winter season water is pumped from a small lake below the mill by an electrically-driven pump; in the summer season it is mainly supplied by a small creek flowing a short distance from the mill.

THE CONCENTRATION AND MARKETING OF CANADIAN MOLYBDENITE.

By H. H. Claudet.

Annual Meeting, Montreal, March, 1917.

The pioneer work on the concentration of molybdenite ores in Canada was initiated by Henry E. Wood, who several years ago treated numerous small shipments from British Columbia in his flotation machine at his testing plant in Denver, Colo. These shipments were increased to a commercial basis during 1915–1916, and up to the present time the main production of molybdenite in Canada has been won by the film-flotation method either on the Wood machine or by employing modifications of the same principle such as in the designs developed at the Dominion ore-testing plant at Ottawa, and also at the mill at Renfrew, Ont., built by the International Molybdenum Company.

Modern methods for the treatment of molybdenite other than film-flotation are confined to oil flotation which, although not yet established in Canada to the same extent, has been and is being successfully applied to molybdenite in different parts of the world. Water concentration can hardly be recognized as an efficient method in itself, although it was formerly used; but the extreme softness of this mineral causes the losses by attrition to be excessive and the thin flakes and fine mineral float away in the tailing. Dry methods of separating by a system of screening can neither be regarded as efficient except perhaps in isolated cases.

Most of the molybdenite ores at present of commercial importance in Canada contain quite a high percentage of iron in the form of pyrite and pyrrhotite and also mica; but they seem to be sufficiently free from such minerals as bismuth, nickel, or wolfram not to require any consideration for the separation and recovery of these minerals as is sometimes the case in other parts of the world as, for example, in Australia. To this extent, Canadian molybdenite ores, containing only one

economic, present a fairly simple problem and as future practice tends towards the smelting of a much lower grade than formerly the mill work should be further simplified thereby. It is clear that the efficiency of a concentrator will be much more easily attained if the concentrate required is only forty per cent to fifty per cent MoS_2 instead of ninety per cent. There seems to be no scientific reason why good practice in smelting molybdenite should not obtain with the former figure, as the only advantage in having a high grade product to treat in the electric furnace for the production of ferro-molybdenum is that there is less gangue matter present and therefore less slag to be taken care of. But as iron is required as a flux, the presence of pyrite or pyrrhotite, which with the prevailing ores would be the principal concomitant in a lower grade molybdenite concentrate, would be acceptable up to a point. To offset the disadvantages of extra freight on transportation of concentrate and increased furnace costs when viewing the smelting of a forty per cent to fifty per cent MoS_2 product as compared with a ninety per cent product, there would be: (1) A simpler design of mill; (2) a more easily operated mill, and, (3) a higher percentage of values recovered. The balance easily seems to be in favour of the lower grade concentrate. I am told on good authority that it is not at all improbable that in the near future the lower grade concentrate will entirely supplant the very high class material. On the face of this it is interesting to note that the requirements of the consumer appear to be a matter of geography. In France they demand over ninety per cent MoS_2, while official enquiries from England are for material of eighty-five per cent or better—although English consumers have frequently been satisfied to take anything over seventy per cent—and there is no difficulty in marketing material as low as sixty per cent MoS_2 in this country.

The system of concentration of any ores depends to some extent on the smelting practice which follows, and as this develops and becomes more efficient the milling is adapted to suit. In some instances low grade base metal ores, difficult to concentrate, but formerly regarded as requiring concentration prior to smelting, are now profitably treated by direct smelting, illustrating the entire elimination of concentration due to

improvements in the smelting process. In considering a comparatively new industry such as the treatment of molybdenite we must be prepared for modifications and changes on similar lines, and it requires no great stretch of imagination to expect that "the ferro-molybdenum of the future will be made from concentrates as low as thirty per cent to forty per cent MoS_2," which is the opinion of an engineer well conversant with the subject.

It may be that the demand still existing for ninety per cent concentrate, or over in some cases, is based on the idea that the more molybdenite present the less room for material of an undesirable nature.

I have dealt rather fully with this question of wide range of preference in grade in the hope that it will be discussed by metallurgists closely associated with the smelting of ferro-molybdenum, who can favour us with direct and reliable information on the subject, and at the same time I trust that we may learn what constitutes undesirable ingredients in a molybdenite furnace charge. Some time ago the writer started to make note of the opinions of various authorities as to undesirable substances, but, as nearly every known element was included in the aggregate, the reliability of at least some of the authorities seemed dubious.

Relating to the different methods of concentration it is too soon to make any reliable comparisons, and in any case each method will probably find special application to certain ores and conditions. Only a general reference to the processes will be made, as there are now standard works describing in detail all the different methods with their machinery. It should be stated that the presence of chalcopyrite in the ore to any extent would be a detriment to all the methods, but fortunately most of the molybdenite deposits now being exploited are practically free from copper. An excess of slime is also undesirable; consequently the crushing operations should be arranged to give as little sliming as possible, whatever treatment is employed.

The film-flotation system now extensively in use and well established as an efficient method requires the ore to be dried in order to slightly oxidize the iron pyrite and other pyrites

without affecting the molybdenite. This is rather a disadvantage as a dry crushing mill presents undesirable features compared with ordinary wet crushing. This method depends on the surface tension of the water, the dry molybdenite floating on the surface forms a tenacious film while the gangue and oxidized sulphides are more easily wetted and sink. The usual practise is to crush the ore to twenty or thirty mesh, and this operation on the dry material produces considerable dust which has to be collected and is probably untreatable, although it is customary to put this impalpable powder into the flotation machines. It is advisable to use a Wilfley table to retreat the tailing after flotation in order to catch the larger pieces of molybdenite which sink owing to their being rounded. The machines are simple and take up very little room and are cheap and easy to make and install. It is necessary to retreat the products in order to get clean concentrates, but separate units, each having a capacity of about six tons per twenty-four hours, can be arranged conveniently in a mill without taking up much floor space.

The Elmore vacuum process has so far been applied more extensively than any other oil flotation method to molybdenite ores, has been producing molybdenite in Norway for several years, and, in 1916, was installed at a molybdenite property in Canada. No sizing, no drying of the ore, and no retreatment is necessary under usual conditions, the preferential flotation being obtained by crushing the ore wet to about ten or twenty mesh and mixing the pulp with a very small quantity of coal oil or other light oil which coats the molybdenite but, not the other sulphides, and in this way the molybdenite separates from the pyrite, pyrrhotite, and also the mica. A settling tank is used to thicken the pulp and to overflow any slimes that may be deleterious and which usually contain between 1% and 2% of the total values. One unit will treat about thirty-five tons in twenty-four hours efficiently, and when everything is properly adjusted only two products are made, namely, a high-grade concentrate and a low-grade tailing. The drawback to this method is that there is more machinery to it and it is of a bulkier nature than the film-flotation plant, the installation requiring a high building; but this concerns the construction work more

than the operating, while the advantages as compared with a mill of the film-flotation type are:—(1) wet crushing instead of dry; (2) no furnace and no fuel required for drying the ore; (3) a marketable concentrate obtained in one treatment; (4) no sizing.

Other flotation methods such as Minerals Separation and Callow do not yet seem to have been established in Canada for the treatment of molybdenite, although I understand Mineral Separation plants are now operating on molybdenite in Norway and Australia, and will be employed shortly in Canada. Of the oil flotation processes usually it has been held that the Elmore process was more suited to the concentration of molybdenite ores than the others, but it is possible that certain ores will favour certain methods.

A description of the film-flotation plant now in use at the Government ore-testing plant at Ottawa will illustrate this type, which is a modification of the Wood machine. The capacity per twenty-four hours is from fifteen to eighteen tons or about 100 tons weekly. There are three flotation machines for roughing, and one cleaner; so the capacity of each machine can be rated at six tons per twenty-four hours and the floor space for each machine is about six feet by five feet. The ore is crushed to $1\frac{1}{2}$ inches in a Blake crusher then elevated to a storage bin and from there fed to rolls set $\frac{1}{2}$ inch apart, and, after the rolls, an automatic sampler cuts out 1-20th. The ore is then elevated to the drier and thence elevated to the Traylor ball mill. It is stated that 30-mesh screens give better results technically but that the 18-mesh is used for commercial reasons, and as with this size the extraction lies between eighty per cent to ninety per cent according to the grade of ore, which varies from $1\frac{1}{4}$ per cent to 4 per cent MoS_2, the work accomplished is very satisfactory—the higher the feed the better the extraction. After passing through the ball mill the ground ore is fed dry to the flotation machines, which make three products: (1) rougher concentrate containing about fifty per cent MoS_2 and sometimes higher with rich ores; (2) middling which is returned to the drier and ball mill to join the circuit for retreatment; (3) tailing to waste, containing about 0.2 per cent MoS_2. The rougher concentrate is dried and sized on a Keedy sizer

into a number of sizes for separate treatment. Usually all the material coarser than 40-mesh will contain seventy per cent MoS_2 or better, and therefore is marketable; the material finer than 40-mesh being retreated separately and the resulting tailing returned to the drier and ball mill for retreatment. The flotation machines require attention and adjusting from time to time to secure best results, but probably no more than other concentrating machines, all of which depend for their efficiency on the care and judgment of the attendants. The chief drawback, as already stated, is that the crushing operations must be performed dry, and so the cost of fuel for drying, which would not be excessive in a centrally located plant, might be prohibitive in remote districts with no available fuel nearby, and where as is the case in respect of the molybdenite mines in Norway, transportation conditions are difficult. The tailing is passed over a 30-mesh Callow screen and the oversize is concentrated on a Wilfley table to catch the coarse pieces of molybdenite, which are returned to the ball mill for retreatment. The type of wet crushing mill, including the Elmore vacuum process similar to those operating in Norway and to the plant installed by the Renfrew Molybdenum Mines near Mount St. Patrick, Ont., consists of a Blake crusher to reduce the ore to about $1\frac{1}{2}$ inch, then an ore-bin. From this bin the ore is fed automatically to a ball mill, then elevated and screened on a separate screen of about 1 mm. opening, the oversize returning to the ball mill. The undersize is sent direct to the Elmore machine, where it first passes through a settling tank the overflow of which carries away some of the slime, and the thickened pulp is mixed with the oil and then separated under the vacuum into the two products, concentrate and tailing. This arrangement gives an extremely simple plant to operate when once all the adjustments are made.

The principal producers at the present time are:—(1) The Canadian Wood Molybdenite Company owning concentrators at their mine at Quyon, Que., and also at Hull, Que. Both this company's plants are dry crushing mills with ball mills and Wood's flotation machines. The entire output from the two mills is sold to the Imperial Munitions Board to be converted into ferro-molybdenum either at Orillia or Belleville. (2) The

International Molybdenum Company, which has treated many different ores at its mill in Renfrew, Ont., the supply from the company's own mines constituting possibly fifteen per cent to twenty per cent of their total requirements. Their plants consists of a dry crushing mill with rolls and their own modification of film-flotation after Wood's method. The concentrate produced in this mill is sent to the company's own smelter at Orillia, Ont. (3) The Renfrew Molybdenum Mines, Limited which treats the output from their mine in their concentrator on the property near Mount St. Patrick, Ont. The plant consists of a wet crushing mill with the Elmore vacuum process. The product from this mill is shipped direct to France.

The mills used as custom plants are situated at Renfrew, Hull, and Ottawa where the Canadian Government ore-testing plant is at the present time employed for that purpose. H. E. Wood's ore-testing plant at Denver, Colo., has been used also as a custom mill more especially for the treatment of the Canadian Wood Molybdenite Company's higher grade ore.

The practice followed is for shippers to sell their ore at a price per unit of MoS_2 varying according to the grade with penalties for copper and bismuth, no payment being made for any molybdic oxide in the ore. Ore is shipped containing as low grade as one per cent MoS_2 up to four per cent or five per cent.

The entire Canadian output for 1916, amounting to 83.69 tons of MoS_2, was produced from ore concentrated at the custom plants situated at Renfrew, Ottawa and Denver, Col., with the exception of what was shipped by the Renfrew Molybdenum Mines.

The following figures will show the distribution of the production for 1916:

(1) Dominion Ore Testing Plant.....recovered 34.84 tons MoS_2.
(2) International Molybdenum Co.'s Renfrew Mill recovered 24.00 " "
(3) H. E. Wood's Denver Plant.....recovered 27.76 " "
(4) Renfrew Molybdenum Mines' Mill.....recovered 1.5 " "

The total of these is a little high owing to some ore from U.S.A. being included in the 2nd and 3rd items.

The Canadian Wood Molybdenite Company shipped during the year:

	5000.0	1. 70% MoS_2 (assuming 80% recovery) = 68.00 tons MoS_2 recovered.	
of which	2180.0	1.74% MoS_2	treated at Dominion plant
	1760.0	1.30 MoS_2	treated at Renfrew plant
& balance	1060.0	2.50 MoS_2 (approximately)	treated at Denver & Quyon
	5000.0		

Over 81 per cent of the total (83.69 tons) was produced from the Canadian Wood Molybdenite Company's mine at Quyon, Que.

The International Molybdenum Company produced thirty tons of ferro-molybdenum at their refinery at Orillia, approximately equivalent to 43.7 tons MoS_2 or 52 per cent of the total MoS_2 produced. These figures are not altogether official, but are as accurate as I have been able to compile, and will be found approximately correct except that the tonnage of the Dominion plant includes some ore treated during Jan. 1917 as a matter of convenience in securing the figures.

1915 was the first year of any commercial output of molybdenite in Canada, the production being 12.75 tons, and this compared with 83.69 tons for 1916 illustrates what substantial progress has been made. From all indications the present output in Canada is not far short of one ton of MoS_2 per diem, or 300 tons per annum; and it is within the bounds of possibility that this amount can be maintained throughout the year, although doubtless there will always be uncertainties to reckon with until the industry has been more firmly established. All of the concentrating plants, with the exception of the Dominion Government plant at Ottawa, have but recently started operations, hence information in the nature of figures and results for publication are scarcely yet available. Meanwhile it is satisfactory to note the progress that has been made during the past year in the development and production of molybdenite in Canada, and as all indications point to fresh exploitations and increased output for 1917, the establishment of the industry on a permanent basis now seems to be assured.

In conclusion it may not be out of place to make some reference to what was accomplished by the staff of the Mines Branch

of the Canadian Department of Mines in fostering the molybdenite industry at the critical time of its inception. After experimental work during 1915, flotation machines were designed in 1916. The ore testing plant was then transformed into a custom mill for the treatment of molybdenite, and, although, as can be imagined a testing plant is not well suited for commercial work, operations have been continuous since June 1916 with highly satisfactory results. Presumably it is the intention to continue regular operations until the mining companies can treat the required amount themselves.

I take this opportunity of acknowledging my indebtedness to Mr. G. C. Mackenzie and his staff for information furnished me relating to the operations of the Government plant at Ottawa, and various other details.

Discussion

Mr. J. B. Tyrrell:—Reference has been made to the property of the Molybdenum Mining & Reduction Company, Limited, on Alice Arm in Northern British Columbia. Last summer, while stopping at Anyox, I had the opportunity of spending a few hours on this property, and Mr. Riel, the Managing Director, very kindly showed me over it. Some of the members of the Institute might like to have my impressions of the difference in character of that ore deposit from those with which they are familiar in eastern Canada. The mining claims in the Alice Arm Group are on a steep and densely wooded mountain side which rises from tide water to a height of several thousand feet. A footpath winds up to them through the forest, and at an elevation of about 1200 feet above sea level it reaches a camp on the bank of a brook which comes tumbling down through a gorge in the mountain. The rock of the country is here a dark gray slate or argillite dipping at a high angle and striking roughly parallel to the shore. In the bottom of the gulch behind the camp, the argillites are intersected by a quartz vein six or eight feet wide which cuts across them in a direction N. 25 E. The vein fills a fault fissure with well-defined walls which are

strongly scored in a horizontal direction indicating a horizontal rather than a vertical movement in the rocks on both sides of the fissure. The vein is irregular both in character and dimensions. In some places it is composed largely of crushed and broken masses of country rock with lateral bands of soft, plastic gouge, while in other places quartz is abundant, and here the vein is usually distinctly laminated parallel to the walls, the darker bands being clearly outlined by the presence in greater or less abundance of fine grained molybdenite and pyrite. Dikes of both acid and basic intrusives are present in the vicinity, and one of the acid dikes was seen running directly down the middle of the banded quartz vein. The gulch has evidently been cut out by the water flowing from the mountain along the line of the vein. Just behind the camp, where the vein, as far as could be seen, is wide, irregular, and composed largely of country rock, with but little quartz, the bottom of the gulch is wide and forms a small basin. At the back of this basin the vein becomes narrower and more regular, and contains a larger proportion of molybdenite, while at the same time the gulch becomes steeper and narrower. At the entrance of this gorge-like portion of the gulch is a talus heap containing many hundreds of tons of angular fragments of the quartz vein-matter containing molybdenite. At this place the sulphides in the vein are chiefly molybdenite and pyrite, but in going northward up the gulch along the course of the vein zinc blende gradually makes its appearance, and at a horizontal distance of about 500 feet from the mouth of the gorge molybdenite has entirely, or almost entirely, disappeared, and has been replaced by zinc-blende and galena, and at the same time the country rock has changed from a dark argillite to a light coloured quartzite, or rock composed of irregular grains of quartz in a sericitic matrix. We have thus here an excellent illustration of a mineral vein changing its character with the character of the country rock from a molybdenite-bearing vein in an argillite country to a zinc-blende and galena-bearing vein in a quartzitic country. The molybdenite differs in physical character from that usually found in the coarsely crystalline rocks of eastern Canada. In this latter part of Canada it commonly occurs in well-defined crystals of larger or

smaller size, but in the property on Alice Arm in British Columbia it conforms more or less closely to the fine-grained or crypto-crystalline character of the vein-quartz with which it is associated, and seems to be in the form of an impalpable powder which is ofter intimately intermixed with the quartz and may be exceedingly difficult to separate from it. I shall not discuss the value of this interesting molybdenum-bearing vein, but I have rather attempted to point out some of its peculiarities which must be taken into account when it comes to be developed intelligently and economically.

Mr. J. A. Dresser:—In listening to Mr. Claudet's paper it has occurred to me that the men handling the production and the concentration of the ore have made in the main greater advancement than those who have been trying to find a supply of ore. The immediate task would seem to be to extend this supply in order that the use of molybdenite may be placed on so sure a basis that its continued use in ordinary times will be assured. This is an excellent problem for the other fellow to undertake. The question seems extremely urgent at this time. Certain deposits of importance have been found in districts that are not known in detail. The exact geological and physical condition under which these occur should be learned and placed on record in order that this knowledge might aid in the intelligent search for other deposits. A detailed study of this question and the prompt publication of information so gained would seem to be of vital importance to the search for and the consequent development of a supply of this ore.

Mr. E. P. Grant (*Communication to the Secretary*).—During the year 1916, the International Molybdenite Co. treated at the new concentrater at Renfrew and at the old plant, a total of 2350 tons of ore containing approximately 60000 lb. of MoS_2. This ore was shipped from properties in Ontario, Quebec, and British Columbia. About half of the total tonnage treated represented customs ores, the remainder being derived from properties operated by the Company. The ores from the different localities are all dissimilar in character, some containing iron sulphides in large amount and some being very micaceous. The British Columbia ores seem to be freer from

these deleterious substances than Ontario or Quebec ores; the flakes, however, are very much smaller, and in some instances, are almost amorphous.

The Renfrew Concentrator—The new concentrator at Renfrew was completed in September, 1916. The process employed is the outcome of experimental operations conducted over a period of about eighteen months.

The location of the plant was formerly a flour mill site, and the concentrator was erected on the hill side thus permitting a difference in elevation of over 40 feet between an 'upper ground floor' and 'a lower ground floor.' The ore enters the mill on the 'upper ground floor.' It goes first to a jaw crusher, thence to a gyratory crusher, and thence either to storage bins or to a rotary drier. The flow up to this point is affected by gravity. The material is then elevated to the top floor (three floors above the upper ground floor) where it is delivered to a large set of rolls and screens, the undersize dropping to a conveyor belt, discharging to storage bins in the flotation department room. The oversize is raised by a bucket elevator to No. 2 rolls and screens, the undersize again being sent to the storage bins in the flotation department. This is in brief the flow-sheet in the crushing department. With ore containing large flakes a fairly good concentrate can be separated by screen adjustment. This concentrate is usually taken to either of the sampling rolls to be cleaned while the undersize goes to the flotation department. The size of material floated varies with the character of the ore. Usually a minus 20-mesh product gives good results. It has been necessary in the case of some ores to crush to minus 60-mesh.

An automatic sampler is placed immediately below the gyratory crusher, and the bulk sample thus produced is reduced to an assay sample in the usual manner.

Flotation Department—The flotation department, housed in a building 90 feet by 30 feet, adjoins the mill. It contains nine flotation machines, 6 feet wide by approximately 18 feet long. The chutes of the storage bins already mentioned, are 12 feet above the ground level and discharge into a 1200-lb. hopper, which travels on rails, and in turn discharges into 2400-lb.

hoppers, feeding the flotation machines. The material discharges from the feed hoppers through adjustable openings to the feed boards of the machines. The concentrate is collected at the end of the machine, dried on steam driers, and is then ready for shipment, or if too low in grade, is given special treatment. The tailing is carried out by waste water to the dump. The Company does not not aim to make an extremely high grade concentrate. The product ranges from 60% to 70% MoS_2, experience having demonstrated that with an average ore it is not commercially advantageous to produce a 90% concentrate. The capacities of the machines vary from 500 lb. to 1500 lb. per hour, according to the character of materials. The finer the material the lower is the capacity.

The mill was designed for 100 tons per day, but at present has a capacity of from 50 to 60 tons. The following record of a few days' run gives an idea of the extraction obtained. The tonnage put through was small, and the extraction a little above normal:

	Lb. Floated	% MoS_2 Heads	% MoS_2 Tails	% Recovery
Oct. 17.....	23400	2.29	.24	90
	22200	1.16	.17	90
" 18.....	15000	1.64	.16	90
	19800	1.66	.21	87
" 19.....	28800	1.35	.26	80
	28600	1.71	.49	71
" 20.....	28200	1.74	.24	86
	20400	1.37	.31	80

Refinery:—The concentrate is shipped to the company's refinery at Orillia. The products of the refinery are ferro-molybdenum, molybdic acid and ammonium molybdate, of commercial quality and also chemically pure. The first stage of the process is to convert the concentrate into crude ammonium molybdate. This is then charged with necessary iron into the electric furnaces. The furnace department has a capacity of two tons of ferro-

molybdenum per twenty-four hours. The furnaces were completed in October, and the first production of ferro-molybdenum was on October 3rd. The ferro contains 70% to 76% Mo., less than 0.1% sulphur, no phosphorus, no copper, and about 3% carbon. All the ferro-molybdenum produced at the present time is supplied directly to the Imperial Munitions Board.

In the chemical department is produced molybdic acid 85% and 99%, and ammonium molybdate. At present only chemically pure products are made, and these are supplied for laboratory purposes in Canada and the United States. Some research work has been done by the company's staff in connection with the production of molybdenum dyes, and also of molybdenum wire.

Production:—The company's production of ferro-molybdenum, molybdic acid, etc., during the year ending December 31st, 1916 had a value of $110,000. These figures do not represent present working conditions, since the regular production of ferro-molybdenum was not commenced until October, 1916. The value of the products being made is now at the rate of over $500,000 per annum, and is increasing rapidly.

The statement may be considered of interest that this company is the only concern in the world that mines and purchases molybdenite ores, operates its own concentrator, uses its own developed process of concentration, refines all the products from its concentrator, and at the same time produces chemically pure reagents.

ELECTRO-DEPOSITION OF ZINC FROM AQUEOUS SOLUTIONS

By E. P. Mathewson, Toronto, Ont.

Annual Meeting, Montreal, March, 1917

Nothing succeeds like success. This applies in metallurgy as in all other vocations; and in recent years we have had striking examples: first, the success of the Smith-Pierce converter using magnesite lining; second, the success of the Mineral Separation process for concentration by flotation; third, the success, obtained by several metallurgists, in producing the highest quality of zinc by electro-deposition from aqueous solutions (notably by Messrs. Laist and Frick at Anaconda and Great Falls, and the Consolidated Mining and Smelting Company of Canada at Trail, B. C.).

The attached bibliography on this subject shows clearly that electro-deposition of zinc is no new thing; but it is decidedly novel to produce zinc in this way, in successful competition with established furnace methods. The Siemens and Halske experts were early in the field and the names of Hoepfner, Engelhardt, Huth and Isherwood are prominent. The Japanese metallurgists attacked the problem about ten years ago, and produced a small quantity of electrolytic zinc prior to the war; but stimulated by the war prices, the Japanese enlarged their operations and I am told are now producing considerable quantities of zinc in three different plants. The General Electric Company has experimented extensively, and has now a plant at Bully Hill, California, for demonstration of the particular process their experts have evolved. Australia and Tasmania have hearkened to the call and are now busy organizing their forces for the production of electrolytic zinc from the vast deposits of ore and tailing available in those countries.

Among so many it is hard to select one name as pre-eminent, but I believe it is only fair to state that to Messrs. Laist and Frick, of the Anaconda Copper Mining Company, deserve the greatest credit in this connection.

Mr. Bretherton in his letter dated October 25th, and published in the *Mining and Scientific Press* under date of December

2nd, sums up very concisely and clearly the conditions in the West prior to the development of the process for successfully depositing zinc electrolytically, and the benefits to be derived from the process referred to. The complex ore question has been, as he states, an unsolved problem for metallurgists for a generation. Now it is practically solved. What under old conditions was waste material, and could not in any sense be considered ore has suddenly become valuable and changed its classification. Even if only 80% of the zinc in complex ores can be recovered successfully by the electrolytic process in its present state of development, the removal of this large percentage of a deleterious element from a complex ore will render the extraction of the remaining more valuable elements quite simple. Western metallurgists now figure that they can treat the so-called complex ores by the new method crediting to the process the values obtained from the sale of silver, gold, copper, and lead and producing the zinc at a price that will compete with the retort process under normal conditions. I believe that even had the war not broken out when it did the electrolytic zinc process would still have been developed. The Directors of the Anaconda Copper Mining Company were so strongly impressed with this belief that they authorized a series of experiments at enormous cost. After these were concluded under the direction of Mr. Frederick Laist, they constructed a plant that produced 10 tons of zinc daily and later increased this to a capacity of 35 tons daily. They then decided to build a modern plant close to the source of power, which is the principal item in the cost of the process, erecting the largest and best equipped electrolytic zinc refinery in the world at Great Falls, Montana. The plant will consist of five units, the first of which was put in operation only a few months ago, while late in December three units were in operation, producing daily 95 tons of zinc, of a purity of 99.92% and over, with physical qualities surpassing anything else on the market. The complete plant having a capacity of from 160 to 175 tons of zinc daily, will be in operation before this paper is published.

During the course of the experiments and during the trials in the test plant at Anaconda the process originally proposed by Messrs. Laist and Frick was much simplified. The process

finally adopted consists essentially of the following steps: First careful roasting of the concentrate at temperatures not exceeding about 730 degrees centigrade; then dissolving the zinc together with a little iron by means of spent electrolyte in Pachuca tanks. A small amount of manganese dioxide is then added to effect the oxidation of the iron, which is then precipitated by means of powdered limestone, bringing down any arsenic or antimony that may be dissolved. These are separated in Oliver filters, and the residue sent to the blast furnaces, while the filtrate, which contains nothing but zinc with a little cadmium and copper, is then treated with zinc dust and again filtered, the filtrate being the pure solution which is sent to the tank room. The anodes are pure lead and the cathodes pure aluminum. The deposition goes on for 48 hours only, when the zinc is stripped from the cathode sheets, then melted into slabs.

A study of the accompanying bibliography will show how many schemes have been tried for obtaining zinc from a complex ore—a problem that has now been solved so simply.

BIBLIOGRAPHY*

1880

LUCKOW, C.

Electrolysis of concentrated zinc solutions.
German Pat. 14256, Apr. 20, 1880.
Abst., Z. Vereins deutscher Ingen., 1882, vol. 26, p. 531.
Wagner's Jahresber. Chem. Technologie 1881, vol. 12 (N.F.), p. 106.

1881

CHANSELLE, J.

Metallurgy of zinc.
Description of the Létrange process.
Bull. Soc. l'Industrie Minérale 1881 (2d Series), vol. 10, pp. 581-95.

LETRANGE, L., Paris.

Application of electricity to the metallurgy of zinc.
Mém. Soc. Ingén. Civils de France, 1881, vol. 34 (II), pp. 602-611.
Abst., Lumière Electrique 1881, vol. 5, pp. 264-66.
Mém. Industrie Belge 1882, vol. 9, p. 13.

LETRANGE, L., Paris.

Process of obtaining zinc from its ores by electricity.
German Patent No. 21775, 1881.
Calamine is dissolved in sulphuric acid and submitted to electrolysis, the sulphuric acid being used again and again. The inventor thinks that with electro-magnetic machines this will be a more economical method than smelting in retorts.

1882

BLAS, C. AND MIEST, E.

Temptative applications of electrolysis. (Recovery of metals from ores.) Louvain and Paris 1882, Gauthier-Villars, Editeurs. (Pamphlet.)
Abst., Berg. u. Hüttenmann. Ztg., 1883, vol. 37 (New Series), pp. 366-67, 378-80.
The processes discovered by Cobley (1878-79), André (1877), Lambotte-Doucet and Létrange (1881), are given in abstract form.

* Prepared by the Library Service Bureau of the United Engineering Society, 29 West 39th Street, New York.

GAUTIER, FERD.

Application of electrolysis to metallurgy. (Lambotte-Doucet process.)
Génie 1882-83, vol. 3, p. 159.

1883

HERRMANN, R. P., Berlin.

Process for the electrolytic precipitation of zinc from dilute zinc sulphate solutions.
German Pat. 24682, of 1883.
Abst., Wagner's Jahresber. Chem. Technologie, 1883, vol. 29, p. 1301.

HERMANN, R. P., Berlin.

Production of zinc from acid-treated ores by electricity.
German Pat. 26091, 1883.
Abst., Wagner's Jahresber. Chem. Technologie, 1884, vol. 30, p. 234.

KILIANI, M.

The working of ores by means of electricity.
Berg. u. Hüttenm. Ztg., 1883, vol. 37 (New Series), pp. 250-52.
Abst., Wagner's Jahresber. Chem. Technologie, 1883, vol. 29, p. 1301.

KOSMANN, B., Breslau.

The production of metallic zinc by electrolysis.
Berg. u. Hüttermann. Ztg., 1883, June 22, vol. 37 (New Series), pp. 287.
Abst., Wagner's Jahresber. Chem. Technologie, 1883, vol. 14 (N.F.), p. 1300.

1884

HERRMANN, R. P., Berlin.

Production of pure zinc by electrolysis.
German Pat. 33107, 1884.
Abst., Wagner's Jahresber. d. Chem. Technologie, 1885, vol. 31, p. 170.

KILIANI, M., Munich.

Production of zinc by electrolysis.
German Pats. 29900 and 32864 of 1884.
Abst., Wagner's Jahresber. Chem. Technologie, 1885, vol. 31, p. 170.

1886

ROSING, B.

On the distillation and electrolytic treatment of zinc-scum.
Zeits. f. Berg.-Hütten- u. Salinenwesen, 1886.
Dingler's Polytechn. Jour., 1887, v. 263, p. 87-94.

In the electrolytic process the zinc-scum is spread out in powder upon a lead plate lying at the bottom of a wooden vat containing a solution of $ZnSO_4$. On passing an electric current through the liquid, the lead plate forming the anode, zinc is deposited from the solution upon the cathode, whilst a corresponding quantity of Zn is dissolved up from the zinc-scum. The action stops when the surface of the particles has become free from Zn. The mass is then heated in a cupellation furnace, and again electrolyzed.

1887

WATT, A., London.

Improvements in the electrolytic treatment of zinc and its ores.
British Pat. 6294, Apr. 29, 1887.

The invention consists in the use as electrolyte of a dilute vegetable acid, preferably commercial acetic acid, or of a solution of zinc or its ores in such a liquid. The anodes may be masses of impure zinc, or, in the extraction of zinc from its ores, may be of C or other insoluble material. The cathodes may be of C, greased or dirty metal, or thin sheets of good Zn. The solution should not be allowed to become too acid, and, in the extraction, must be produced and the acid formed in the bath allowed to take up more ore, though the solution is preferably kept diluted with water to a density of 1.15 or 1.16. Calamine may be crushed, sifted, and treated directly. Blende must first be roasted. An excess of metallic impurities in the solution may be precipitated by sulphuretted hydrogen. The advantages claimed are better and more regular deposits and the production of very little gas.

1888

BURGHARDT, C. A.

Improvements in the production of pure zinc.
British Pat. 18440, Dec. 18, 1888. German Pat. 49682, May 22, 1889.

Impure zinc is employed as anode in an electrolytic cell, the deposit of pure zinc being received on a suitable cathode. Zincate of Na or K is used as electrolyte, and may be obtained by charging the cell with caustic alkali, and passing a current of electricity until pure zinc is deposited.

WATT, A., London.

Electrolytic treatment of zinc.
Lumière Electr. 1888, vol. 29, p. 144.

WATT, A., London.

Improvements in the treatment of zinc and its ores.
British Pat. 3369, Mar. 5, 1888.

When lead is present in the ores, it interferes with the proper deposit of the Zn. The author claims the idea of adding to the bath H2SO4, which precipitates the Pb as PbSO4. Impure zinc, containing Pb, can be used as the anode in this case. When Pb is present in large quantity, the ore or metal is treated first with acetic acid, which dissolves the lead, and afterwards with dilute sulphuric acid. The Pb is precipitated from the acetate solution by scrap zinc, and the solution of acetate of zinc so produced added to the solution of sulphate to make up the bath for electrolysis.

1891

ANONYMOUS.

A new electrolytic separation of zinc (Kosmann and Lange-Brieg processes).
Electr. Rev. & Telgr. Jour., 1891, vol. 28, p. 749.

1892

ANONYMOUS (G. R.).

Electrometallurgical extraction of zinc by the Nahnsen process.
Lumière Electrique, 1892, vol. 44, p. 376.

CASSEL, G. E., and KJELLIN, F. A.

An improved method of producing zinc from blende by means of electrolysis.
British Pat. 21193, Nov. 22, 1892, German Pat. 67703, Aug. 18, 1892.

Zinc is produced by blende by means of electrolysis, the ore being first converted into a soluble salt of zinc, which is electrolyzed in a bath in which the cathode consists of zinc or other suitable metal which is not corroded in the process, and is separated from the anode by a porous partition. Iron, or some analogous metal, is used for the anode, and the electrolyte round the cathode consists of a soluble Zn-salt, and that round the anode of a solution of a salt the positive component of which consists of the metal used as anode, and the negatives of the same acid as that contained in the salt of zinc to be electrolyzed. If the acid set free during the operation forms insoluble salts with the anode, the porous partition may be dispensed with and the cathode and anode are both surrounded by the salt of zinc to be electrolyzed.

NAHNSEN.

Contribution to the electrolysis of zinc salts.
Berg. u. Hüttenmann. Ztg., 1891, Oct. 23, vol. 50, No. 43, pp. 393-97.

NATHUSEN, G., Cologne.

Improvements in the electro-metallurgical extraction of zinc.
British Pat. No. 2913, Feb. 15, 1892.
Abst., J. Soc. Chem. Ind., 1892, v. 11, p. 535.

NAHUSEN, Cologne.

Improvements in the methods of purifying electrolytes containing zinc, relating to or connected with the electro-metallurgic production of zinc.
British Pat. 2225, Feb. 5, 1892.
Abst., J. Soc. Chem. Ind., 1892, v. 11, p. 535.

TOMMASI, D., Paris.

Process and apparatus for the extraction, separation and refining of metals by electrolysis.
British Pat. 16893, May 21, 1892.
Abst., J. Soc. Chem. Ind., 1893, vol. 12, p. 768.

TOMMASI, D., Paris.

Process and apparatus for the extraction, separation and refining of metals by electrolysis.
British Pat. 16892, Apr. 1, 1892.
Abst., J. Soc. Chem. Ind., 1893, vol. 12, p. 768.

1893

ANONYMOUS.

Recent progress in the electrometallurgy of zinc.
Dingler's Polytechn. Jour., 1893, vol. 288, pp. 258-63.

COEHN.

Electrolytic metal deposition.
German Pat. 75482, July 1, 1893.

In all such cases where the evolution of H at the cathode is the reason for the loose to spongy form of the metallic deposits, the application of an intermittent current has proved advantageous. The number of current interruptions required is to be determined separately in accordance with the quantity of gas evolved with constant current. In the electrolysis of zinc sulphate, for instance, a compact, strong deposit was obtained with about 50 interruptions per minute and a current density of 50 amp. per sq. m. There need actually not be real interruption of the current, but a constant switching from one bath to the other,

the switch being operated by a small electromotor or a small turbine. A rotating drum consists of one-half of its periphery of insulating, solid material. In contact with this drum, the metallic axis of which is connected with the current generator, are two brushes, one of which leads to the one series of baths, the other to the other series.

1894

HOEPFNER, C.

Improvements in the treatment of calamine and similar minerals or compounds for the extraction of zinc.
British Pat. 11724, June 16, 1894.
Abst., J. Soc. Chem. Ind., 1895, vol. 14, p. 581.

MYLIUS AND FROMM.

The precipitation of metals from dilute solutions. (II.)
Ber. d. deuts. Chem. Gesellsch., 1894, vol. 27, pp. 645-51.
Abst., J. Chem. Soc. (London), 1894, vol. 66, Part II, pp. 235-37.

1895

URGHARDT, C. A. AND RIGGS, G.

Improvements in a process and apparatus for the electrolytic production of zinc from ores of zinc, and metallic Zn and metallic Cu from Zn ores containing Cu, and the production of O gas in the cases specified.
British Pat. 22732, Nov. 28, 1895.

The ores which have been thoroughly oxidized by roasting, are ground and treated with a concentrated solution of either the normal carbonate, the sesquicarbonate, or the bicarbonate or ammonium, or a mixture of these; or a solution of ammonia gas in water may be employed, but in the latter case no oxygen is evolved in the latter stages of the process. The bath is covered with a layer of paraffin or other mineral oil of higher flashing point and low viscosity, to prevent loss of ammonia, and a mechanical stirrer is set in motion. When all the oxides have been dissolved, the clear liquor is drawn off into another tank, and again covered with a layer of oil. The solution is freed from any Fe it may contain by treatment, at about 40 degrees C., with a certain proportion of hydrated oxide of tin, and the clear liquor is again drawn off into another tank. If Cu be present, it is recovered by immersing in the bath plates of Zn, or preferably, of Zn in contact with Sn, and the Cu deposited on the Zn strips easily, and is entirely free from oxides, owing to the layer of oil. The solution, free from Cu, is then electrolyzed in a bath covered with a layer of paraffin and provided with sheet zinc cathodes, and tin or lead anodes. If a solution of the carbonates has been employed, the oxygen given off at the positive pole is collected and washed.

HASSE, E.

Zinc electrolysis with aqueous electrolytes.
Berg. u. Hüttenmann. Ztg., 1895, vol. 54, No. 50, pp. 431-35, No. 51, pp. 439-443. (Fig. 1-12, Plate VII.)
A patent review.

HOEPFNER, C., Giessen, Germany.

Improvements in or relating to electrolytic or galvanic processes or cells.
British Pat. 17745, Sept. 23, 1905.
Abst., J. Soc. Chem. Ind., 1896, vol. 15, p. 816.

LORENZ, R.

Method of extracting zinc and lead from mixed ores.
Zeits. f. Elektrochem., 1895-96, vol. 2, pp. 318-337.
Abst., J. Soc. Chem. Ind., 1896, vol. 15, pp. 278-80.

MYLIUS, F., AND FROMM, O.

Experiments on the production of pure zinc.
Zeitschrift f. anorg. Chem., 1895, v. 9, p. 144-77.
Ber., 1895, v. 28, pp. 1563-75.

The authors commenced by analyzing a number of samples of "commercially pure" metal. 100 grams were used in the process, and the basis of their analytical method was a preliminary fractional precipitation with ammonium sulphide.

The impurities are chiefly Pb, Cd and Fe, *not* arsenic. The following quantities were estimated in 100000 parts on 8 samples of commercially pure zinc from various sources, seven of them reputed pure zinc from leading manufacturers, and in the table below numbered 1 to 7:

	Pb	Cd	Fe
1	72	111	15
2	30	58	11
3	5	23	2
4	5	16	14
5	10	18	2
6	21	5	36
7	11	99	4
Electrolytic zinc, commercial	58	39	17

Various processes for the purification of impure zinc were investigated as follows:

I. Dry processes: Recrystallization of the metal reduced the lead—impurity from 0.67% to 0.02% after five repetitions; the Cd fell from 1.27 to 0.36% in four recrystallizations The iron, on the other hand, increased from 0.92 to 3.2% in two crystallizations. This process can scarcely be recommended.

II. Wet processes (Electrolytic):

(1) By preparing the metal from solutions of purified salts with insoluble anodes; (2) by electrolytic purification by means of soluble anodes of zinc.

In all cases purified zinc sulphate was used. Iron was eliminated by heating the solution with zinc oxide in presence of hydrogen peroxide. After warming and filtering, a solution free from iron is obtained. For the elimination of other impurities the following methods are recommended:

(a) Recrystallization of the sulphate from solution in dilute alcohol. (b) Decomposition of zinc ethyl with sulphuric acid. (c) Electrolysis of the solution with Pt- electrodes and weak currents, keeping the solution neutralized by means of zinc oxide. Pb, Cd and Fe11 are separated at the cathode; PbO_2 and Fe_2O_3 at the anode. (d) Treatment of $ZnSO_4$ solution with purified zinc. (e) Treatment of porous zinc (electrolytic) with less than the equivalent of sulphuric acid in dilute solution.

A pure zinc oxide, required in the above treatments, may be prepared from the nitrate by fractional precipitation with ammonium sulphide.

The main issue of the investigation is that the purest zinc is obtained by repeated electrolytic separation of the metal in the spongy form from basic solutions of the sulphate. The impurities do not exceed 1 part per 100,000. As a final purification (elimination of non-metallic elements, dissolved gases, etc.) the authors recommend distillation in vacuo.

SIEMENS & HALSKE, Berlin.

Improvements in the electrolytic production of zinc from its ores, and apparatus therefor. British Pat. 13434, July 12, 1895.

The improved process for obtaining zinc by electrolysis consists in first roasting the zinc ore, then lixiviating the same preferably by means of sulphuric acid, and, after separation therefrom of the greater part of the impurities by known means, subjecting the same to electrolytic action by means of zinc cathodes and anodes of C, Pb or other bodies not readily attacked by oxygen whilst at the same time air under pressure is forced in at the lower part of the cells, so as to produce a rapid circulation of the electrolyte, by which means it is claimed that the formation of zinc sponge is prevented. The zinc solution is preferably concentrated to about 10 degrees B., and subjected to preparatory electrolysis until it contains 5-10 grms. of free sulphuric acid per litre. The cathodes are then cleaned from any sponge that may have been formed, and subsequently the special regular process commences, in which the acid formed in the depositing vats is fixed by dissolving zinc oxide. The pressure employed is 1.3 cubic meters of air per hour for electrode plates of about 1 meter length and 0.7 m. width, and a current density of about 50 amperes to the square meter. Partitions of wood or other neutral material extend from near the surface of the liquid in the cell to near the bottom of the vat, so as to form separate compartments, and perforated pipes passing near the lower edge of the partition admit the air under pressure.

1896

ANDREOLI.

Electrodeposition of zinc by the Cowper-Coles process.
L'Electricien, 1896, vol. 12, p. 99.

ANONYMOUS.

Simultaneous electrolytic deposition of zinc and lead from aqueous solutions.
Oester, Zeits. Berg. u. Hüttenwesen, 1896, col. 44, pp. 13-14.
Elektrotechn. Znzeiger, 1895, vol. 12, pp. 1869-71.

HOEPFNER, C.

Improvements in and relating to the treatment of ores or materials containing Zn, Pb and Ag.
British Pat. 8328, Apr. 26, 1896.
Abst., J. Soc. Chem. Ind., 1896, vol. 15, p. 458.

NEUMANN.

Electrolytic extraction of zinc.
Chem. Ztg., 1896, vol. 20, pp. 36-38.
Abst., J. Soc. Chem. Ind., 1896, vol. 15, pp. 280-81.

DIEFFENBACH, O., Darnistadt.

Improvements in the electrolytic production of zinc and alkalis.
British Pat. 25804, Nov. 16, 1896.

The electrolytic vessel is divided into three parts by two diaphragms, the anode and cathode compartments containing respectively sodium sulphate and an alkaline solution of Zn, while the centre compartment contains Na_2SO_4, or a solution of alkaline chloride or dilute alkaline lye. In the first case, on the passage of the current, zinc is deposited in the cathode compartment, and the strength of the solution in zinc may be maintained by heating it together with ZnO, Zn $(OH)_2$, or even metallic zinc. Simultaneously with the separation of Zn, NaOH is formed, which passes into the centre compartment instead of

being neutralized by the free acid formed in the anode compartment, as is the case when only one diaphragm is employed. The anode solution is kept constantly circulating, and the acid is neutralized with lime, etc., as soon as it is formed. The solution in the centre compartment is regularly drawn off, and the NaOH there formed separated by evaporation from the Na2SO4, which crystallizes out.

1898

ANONYMOUS.

Electrolytic zinc.
Engineering, 1898, Jan. 21, v. 65, p. 85-6.

ANONYMOUS.

Cowper-Coles electrolytic method for the extraction of zinc from refractory ores.
Electrical Rev. (London), 1898, v. 43, No. 1082, pp. 265-67. Ills.

ANONYMOUS.

The combination of the desilverization process of lead by means of zinc with the electrolytic production of zinc and silver according to the Haase process.
Berg. u. Hüttenm. Ztg., 1898, vol. 57, p. 46-8.

ANONYMOUS.

The electrolysis of zinc solutions at Broken Hill.
L'Electricien, 1898, vol. 16, (2d Series) pp. 103-05.

FOERSTER, F. AND GUNTER, O.

Electrolysis of zinc chloride solutions and the nature of spongy zinc.
Zeits. f. Elektrochem., 1898, v. 5 (1), pp. 16-23.

Starting with the assumption that the deposition of spongy zinc is caused by the presence of precipitated zinc oxide on the cathode, the authors have made some experiments which tend to confirm this view.

A zinc cathode was suspended in a perfectly neutral purified zinc chloride solution containing 54.6 grams of Zn per litre, between two anodes of pure electrolytic zinc. With a current density of 1.4 amperes per sq. dom. a smooth compact deposit of zinc was obtained on the cathode for the first six or seven hours, but after 20 hours spongy zinc was being deposited, and the solution was turbid from the presence of suspended basic zinc chloride.

On filtering off the precipitate and inserting a new cathode, a good deposit was obtained during the first few hours. The electrolyte then again became turbid, and, at the same time, the deposit showed signs of becoming spongy. Schnabel (Handbuch der Metallhüttenkunde, 2, p. 114) states that the formation of spongy zinc may be prevented by the addition of Cl or bleaching powder to the solution. The authors, therefore, tried the method adopted by Mylius and Fromm for keeping a solution of zinc sulphate constantly acid. A small quantity of free Cl was continuously evolved in the cell from two small Pt-anodes, which were connected with the positive pole of two accumulators, whilst the negative pole was attached to the large zinc plate, which thus served as cathode for both pairs of anodes. The formation of spongy zinc, however, could not be prevented in neutral solutions by this arrangement.

The O of the air easily oxides zinc in presence of ZnCl2, and a certain quantity of oxid formed is dissolved by the solution forming basic zinc chloride. Deposition of spongy zinc only occurs when the excess of oxide begins to render the electrolyte turbid, and if this precipitation is prevented by the periodic addition of hydrochloric acid, good deposits of zinc can be obtained. The authors, therefore, recommend that the electrolyte should contain a quantity of free HCl corresponding to 1-20 to 1-30 normal solution. The acid may be increased until it represents 1-10 normal acid, but it should not be allowed to fall below 1-200 to 1-300 normal.

If this slightly acid electrolyte is used in conjunction with the arrangement for liberating free Cl, still better deposits of zinc are obtained. When the solution contains 0.2 gm. of free Cl per litre, the H evolved at the cathode does not remain sticking to the electrode long enough to impair the surface of the deposit.

Although the main body of the electrolyte remains slightly acid, it sometimes happens that lead (spongy) deposits of zinc are obtained. This is accounted for by the fact that the layer of electrolyte in contact with the cathode becomes deprived of its free acid, but this difficulty may be overcome by slightly agitating or circulating the electrolyte.

On substituting anodes of commercial zinc for the pure Zn- anodes, the Zn deposited on the cathode was formed to be entirely free from arsenic.

When a 40% boiling solution of ZnCl2 is saturated with zinc oxide (the Cd subsequently removed by treatment with pure zinc), electrolyzed at 10° C, with insoluble (Pt) anodes, good deposits of zinc are obtained. The quantity of zinc thus deposited, when compared with the amount of copper deposited by the same current in a copper voltmeter was found to be in good agreement with Faraday's Law. Ashcroft, on electrolyzing a similar solution, found that the zinc deposited in the first two hours was 49%, during the next two hours 37%, and during the third two hours 14% in excess of that required by Faraday's Law, whilst in the last two hours the theoretical amount of deposited. The authors conclude by discussing the nature of spongy zinc and the effect of various oxidizing and reducing agents to the electrolyte. The addition of a perfectly neutral solution of ammonium persulfate to a neutral solution of zinc chloride furnished good deposits of zinc on electrolysis, whilst on the addition of 0.025% of H2O2, the formation of spongy zinc was observed after electrolyzing 10 minutes.

STRZOVA, W., Zalenze.
A process for the electrolytic extraction of zinc, and other metals from ores or waste containing them.
British Pat. 24307, Nov. 17, 1898. German Pat. 118291, Oct. 19, 1898.
A process for the electrolytic extraction of Zn from their ores or waste containing them; in which the pulverized ore, such as calamine, blende and the like, or waste containing Zn, such as zinc ash, etc., is exposed to the action of an electric current directly in an alkaline bath (caustic soda), without any preliminary chemical treatment whereby the zinc and also the other metals present are at once in such bath metallically separted at the cathode.

INGALLS, W. R.
The Ashcroft Process for the treatment of mixed lead and zinc ores.
Engineering and Mining Journal, Oct. 22nd, 1898.

1899

ANONYMOUS.
The present status of the electrolytic zinc industry.
Elektrochem. Zeitschr., 1899, vol. 6, No. 5, pp. 99-100.

COWPER-COLES, Sher.
The electrolytic treatment of complex sulphide ores.
Electrician 1899, vol. 44, pp. 81-83, 111-13, 157-59.

COWPER-COLES, Sher.
Some notes on the electro deposition of zinc under pressure.
Electrician (London), 1899, vol. 44, pp. 183.

FOERSTER, F. and GÜNTHER, O.
Further contribution to the nature of zinc sponge.
Zeitscher. f. Elektrochemie, 1899, vol. 6, No. 22, pp. 301-3.
Berg. u. Hüttenmann. Zstcher., 1899, vol. 58, pp. 17-9.
Chem. News, 1899, vol. 80, pp. 209-10.

KERSHAW, J. B. C.
The electrolytic methods of zinc extraction.
Electrical Rev., 1899, July 7, vol. 45, No. 1128, pp. 36-37; 43.

1900

CLANCY, J. C., and MARSLAND, L. W.
Process for the elimination of zinc from sulphide ores, and the extraction and recovery of lead, silver, gold and other metals therefrom, and from other sulphide ores.
British Pat. 4039, Nov. 26, 1900.
Abst., J. Soc. Chem. Ind., 1901, vol. 20, p. 481.

HOEPFNER, C.
Process of extracting zinc or other metals from their ores.
British Pat. 15873, 6-IX-1900.
German Pat. 155065, I-IV-1902.
U. S. Pat. 704641, July 15, 1902.
Abst., J. Soc. Chem. Ind., 1902, vol. 12, p. 1081.

KELLNER, C., Vienna.
Extraction of zinc from waste products.
British Pat. 7028, Apr. 14, 1900. U. S. Pat. 690295, Dec. 31, 1901.
Abst., J. Soc. Chem. Ind., 1901, vol. 20, p. 367.

RONTSCHEWSKY, J.
Electrolytic production of zinc with insoluble anodes and simultaneous recovery of the anode material.
Zeits. f. Elektrochemie, 1900, vol. 7, No. 2, pp. 21-5.

WORSEY, J. W., and LANCASHIRE, J. H.
Process for the treatment of complex ores (Pb, Zn, S, etc.).
British Pat. 3668, Feb. 24, 1900.
German Pat. 135056, Dec. 22, 1900.
Abst., J. Soc. Chem. Ind., 1901, vol. 20, p. 367.

1901

MOND, L., London.
Method of obtaining Zn by electrolysis and apparatus for that purpose.
British Pat. 5764, Mar. 19, 1901. German Pat. 134862, Aug. 9, 1901.
In order to obtain zinc in a solid, instead of a spongy form, when deposited electrolytically, the cathodes are made cylindrical, two or more drums being pressed against each other by springs. The cylinders, preferably of slightly different diameters (to avoid synchronism during rotation), are caused to revolve whilst the current is passing, and an axial reciprocating motion is also imparted to them, so that the zinc immediately after it is deposited on the cylinders becomes compressed while passing the successive lines of contact

of the cylinder with those next adjoining. The cylinders may be rotated separately, or one or more alone be rotated, the others being rotated by frictional contact with them. Similarly the traversing action need not be imparted to each of the cylinders. The anodes are preferably shaped with corrugations, corresponding to the contour of the cathrode drums, so that the inter-electrode spaces are fairly uniform. Different arrangements of vertical and horizontal cathode cylinders are described and protected.

PETERS, FR.

Principles and present status of the electrolytic production of zinc.
Berg. u. Hüttenmann, Ztg., 1901, vol. 60, No. 49, pp. 587-92; No. 50, p. 599-604.

TOSSIZZA, C. J., Paris.

Electro-metallurgical treatment of zinc ores and especially blende.
British Pat. 14825, July 20, 1901. U. S. Pat. 703857, July 1, 1902.

This process is a modification of that described in British Patent 14751, 1901, which is applicable to sulfide ores in general, but is not suitable to those of Zn. With Zn ores, the electrolysis is carried out in two separate operations. In the first, a zinc salt is electrolyzed, using soluble anodes, at a voltage below that required for decomposing water; and in the second, the salt formed at the anodes by the previous operation is decomposed, and depolarization and reduction of voltage are ensured by maintaining around the insoluble anodes a solution of sulphurous acid obtained from roasting of the blende, the two operations together effecting, it is stated, electrolysis of zinc sulphate, and producing non-spongy zinc, with an E.M.F. of about 1.75 volt, or less than that required for a direct operation.

1902

CLANCY, J. C., and MARSLAND, L. W.

Process for the treatment of zinc sulphide ores.
U.S. Pat. 715023, Dec. 2, 1902.
Abst., J. Soc. Chem. Ind., 1903, vol. 22, p. 33.

PETERS, F.

Progress in electrometallurgy.
A process for the electrolytic production of zinc from its ores
Berg. u. Hüttenm. Ztschr., 1902, vol. 61, p. 528.

SADTLER, S. S.

A method of electrolytic production of zinc from its ores.
Trans. Amer. Electrochemical Soc., 1902, vol. 1, pp. 141-44.
Discussion pp. 144-46.

SMITH, E. F.

Observations on the electrolytic precipitation of zinc and of copper.
J. Amer. Chem. Soc., 1902, vol. 24, pp. 1073-76.

1903

ANONYMOUS.

Electrolytic extraction of zinc.
Génie Civil, 1903, vol. 43, pp. 124-5.

AMBERG, R.

Electrolysis of alkaline solutions of zinc.
Ber. Chem. Gesells., 1903, vol. 36, No. 11, p. 2489-94.
Abst., J. Soc. Chem. Ind., 1903, vol. 22, p. 955.

GUENTHER, E.

Electrolytic zinc extraction by the Hoepfner process.
Engng. & Min. Jour., 1903, May 16, vol. 75, p. 750.

HOEPFNER, C.

Process of obtaining lead or other metals from ores or matter.
U.S. Pat. 735098, Aug. 4, 1903.
Abst., J. Soc. Chem. Ind., 1903, vol. 22, p. 1001.

HOLLARD et BERTIAUX.

Influence of gases upon the separation of metals by electrolysis; separation of nickel and zinc.
Compt.-Rendus, 1903, vol. 137, pp. 853-5.

KAISER, K.

Process for obtaining zinc by electrolysis.
French Pat. 333773, July 11, 1903. British Pat. 15420, 11-VII, 1903.
Abst., J. Soc. Chem. Ind., 1903, vol. 22, p. 1356.

SADTLER, S. S.

A method for the electrolytic extraction of zinc from its ores.
Elecktrochem. Zeitschrift, 1903-04, vol. 10, pp. 1-3.

STRZODA, W.

A few facts regarding the electrolytic production of metals, in the wet way, with particular reference to the production of zinc.
Chem. Ztg., 1903, vol. 27-11, p. 741.
Abst., J. Soc. Chem. Ind., 1903, vol. 22, p. 955.

1904

BRANDHORST, H.

Contributions to the metallurgy of zinc.
Z. Agnew. Chem., 1904, vol. 17, pp. 514-517.
Abst., J. Soc. Chem. Ind., 1904, vol. 23, p. 491.

BURLEIGH, CH. H.

Electrolytic production of zinc and lead from complex sulphide ores
Electrochemical Industry, 1904, Sept., vol. 2, No. 9, pp. 355-57.

GUNTHER, E.

Die Darstellung des Zinks auf elektrolytischen Wege. Book.
W. Knapp, Halle. Price 10 Marks.

HESS, J.

Electrolytic production of zinc.
Zeitschr. f. Elektrochem., 1904, vol. 10, pp. 686-93; 764-68.
A complete review up to 1904.

KERSHAW, J. B. C.

Notes on progress made in Europe and America in industrial electro-chemistry and electro-metallurgy.
Electr. Rev. (N.Y.). 1904, vol. 44, p. 325.

LASZEZNSKI.

Electrolytic deposition of zinc and copper from ores.
U.S. Pat. 757817, Apr. 19, 1904.
Metallurgie, 1904, vol. 1, pp. 439-441.

SWINBURNE, J.

Chlorine in metallurgy.
Paper read at the International Electrical Congress of St. Louis (Mo.), 1904.
Abst., Electrochem. Industry, 1904, vol. II, No. 10, p. 404.

1905

DECKER MANUFACTURING CO.

Electrolytic recovery of Zn, Mg and other substances in acid or neutral solution.
French Pat. 352029, Mar. 3, 1905. German Pat. 184516, Mar. 1, 1905.
Abst., Z. Agnew, Chemie, 1908, v. 21, p. 320.

The invention relates to an apparatus comprising a tub containing the electrolytic solution, the vessel being closed by a partition pierced with one or more openings, in each of which is lodged a trough. The ends of each trough are furnished with projecting pieces fitting into the ends of the trough below, and the sides are sloping, the bottom being inclined towards the outside. The troughs contain Hg, which closes the openings of the partition, and the troughs are all connected together electrically. The troughs are also provided with overflow pipes, so that the Hg mercury poured into the upper trough passes down successively from one trough to the next below.

FERCHLAND, P.

Electrodes for electrolytic purposes.
British Pat. 24806, Nov. 5, 1906. German Pat. 206329, Nov. 10, 1905.
French Pat. 371245, Nov. 8, 1906. U.S. Pat. 900502, Oct. 6, 1908.

Lead peroxide (PbO_2) is deposited electrolytically in a compact form from solutions of lead salts on all sides of suitable conductors, the latter being rotated around their own axis during deposition; the conductors are afterwards removed.

LANGBEIN, DR. G., & CO.

New Process for the electro-deposition of zinc and its alloys.
French Pat. 358622, Oct. 18, 1905.

The process of electro-deposition of Zn upon metallic objects is improved by the addition of urea to the bath. Double salts which are readily soluble are formed, and a high current density can be maintained without resulting in the production of a spongy deposit.

PETERS, FR.

Contribution to the electrometallurgy of zinc.
Glückauf, 1905, Dec. 2, vol. 41, pp. 1496-1502.

TOSSIZZA, C. J., Paris.

Electrolytic extraction of zinc from its ores.
French Pat. 370803, Dec. 30, 1905. German Pat. 182736, Dec. 28, 1905.

Salts of Zn are electrolyzed, using insoluble anodes, in a vat divided into two compartments by a porous diaphragm. The solution in the anode compartment is saturated with SO2 to act as a depolarizer, in order to permit electrolysis to be carried on with a low and constant voltage. Around the cathode is maintained a liquid charged with substance which will neutralize the free H2SO4 produced at the anodes. Evolution of H at the cathode is thus avoided, and good deposits of Zn are obtained.

1906

ENGELHARDT, V. Assigner to Siemens & Halske Akt.-Ges.

Electrolysis of zinc sulphate.
U.S. Pat. 831843, Sept. 25, 1906.
Abst., Electrochem. & Metallurg. Industry, 1906, vol. 4, No. 10, pp. 416-17.
Elektrochem. Zeitschrift, 1906-07, vol. 13, p. 204.

PRICE, J. S., and JUDGE, G. H. B.

Electrolytic deposition of zinc, using a rotating electrode.
Trans. Faraday Society, June 12, 1906.

The authors studied the quantitative determination of zinc in zinc sulphate solutions, using the rotating electrodes recommended by Perkin (J. Soc. Chem. Ind., 1903, p. 1065, fig. 3). The solution is contained in a tap-funnel of about 100 cc. capacity. Quantitative results are obtained if the solution is kept cool, even when the zinc sulphate solution is initially slightly acid. The addition of sodium acetate and Na2SO4 was tried, and in the latter case good results were obtained. A modified form of Kollock and Smith's apparatus (J. Soc. Chem. Ind., 1905, v. 24, p. 1127) is also described, in which a mercury cathode and rotating anode are employed. Here again a cylindrical vessel provided with a tap is used to contain the electrolyte, and, after the completion of the experiment the Zn- amalgam can be run off through the tap, and subsequently weighed.

1907

ANONYMOUS.

Electrolysis of zinc sulphate.
Rev. Indust., 1907, vol. 38, p. 78-9.
Reference is made to Price and Judge's work as published in the Faraday Soc. Trans.

ENGELHARDT, V., and HUTH, M.

Charlottenburg. Assignor to Siemens & Halske A-C. Berlin.
Electrolytic winning of zinc in compact form from solutions of zinc sulphate without the use of a diaphragm.
French Pat. 375140, Feb. 28, 1907. U.S. Pat. 935250, Sept. 28, 1909.

Under ordinary conditions it is found that Zn electrolytically deposited from sulphate solutions redissolves so soon as the electrolyte contains any considerable amount of free acid. This is specially disadvantageous, since the metal obtained from metal solutions is well crystallized and free from oxide. The inventors have found that the solubility is caused by the presence of metallic impurities, which dissolve in minute quantity from the Pt, Pb, or C-anodes previously employed, and become deposited on the cathode. The present invention consists in using anodes of PbO2, either in the form of compact pieces without metallic support, or with a support of some insulating material, which is not attacked by the acid solution, and which serves to augment the mechanical compactness of the PbO2.

GOLDSCHMIDT, R.

Notes on the electrolytic deposition of zinc from aqueous solutions of zinc fluosilicate.
Revue d'Electrochimie et d'Electrométallurgie 1907, vol. 1, pp. 276-77.

PAWECK, H. Vienna.

Electrodeposition of zinc.
British Pat. 8562, Apr. 12, 1907. French Pat. 386678, Apr. 15, 1907.

Zinc is deposited electrolycically from a bath containing one or more salts of conductive salts or other substances. The liquid is maintained neutral, or preferably slightly acid, by intermittent and continuous regeneration, and contains 250 grms. of Zn-salt per litre of water, or more, up to the point of saturation. The continuous regeneration of acid is affected by the addition of substances which yield H-ions in aqueous solution, so that these need only be added at long intervals. Substances such as boric and phosphoric acid and the like, may also be added, these being ionized to a smaller degree. These compounds act as a reserve, and come into action on the exhaustion of the hydrogen ions in the substance used for continuous regeneration.

PRICE, T. S. and DUDGE, G. H. B.

The electrolytic deposition of zinc while using rotating electrodes.
Transl. from advance copy in Elektrochem. Zeitschr. 1907, vol. 13, pp. 233-37 and vol. 14, pp. 113-6.

SNOWDON,

Electrodeposition of zinc.
Amer. Electrochem. Soc. Trans. 1907, vol. 11, pp. 121-134.
Abst., Chem. Ztg., 1907, vol. 31, p. 739.

1908

BECHI, and RUCKER, R. W.

Process for treatment of complex zinciferous ores.
Mines and Minerals, 1908, vol. 28, p. 298.

PRICE, T. SLATER.

The electrolytic deposition of zinc, using rotating electrodes. (II)
Chem. News, 1908, Feb. 21, vol. 97, pp. 89-90; 99-102.

SIEMENS and HALSKE AKT.—Ges., Berlin.

Process for removing iron from zinc sulphate solutions to be used for the electrolytic separation of zinc by means of peroxide and zinc oxide.
German Pat. 213004, Apr. 12, 1908.

MnOI is used for the oxidation of the Fe″ contained in ZnSO4 solutions, with a view to the subsequent precipitation of the Fe with ZnO. The MnO2 is regenerated during the electrolytic separation of the zinc from the purified solution.

SANNA, E.

Process for the utilization of spongy zinc obtained electrolytically.
German Pat. 217632, Oct. 6, 1908. British Pat. 24631, Oct. 26, 1909.

The spongy zinc is washed in a solution of NaNO3, and then while still moist is heated to 100-150° C. and simultaneously exposed to a pressure of about 1000 kilos, per sq. cm. It is stated that dense metallic Zn is readily obtained in this way.

WILLIAMS, J. R. and BRADLEY, H. W. and B., Sheffield.

Process for obtaining zinc or copper from complex ores or the like.
British Pat. 26711, Dec. 9, 1908. Austrian Pat. 6141, Aug. 9, 1909.
Abst., J. Soc. Chem. Ind., 1910, vol. 29, p. 160.

1909

HUTH, M., Assignor to Siemens and Halske A.-G., Berlin.

Apparatus for the electrolysis of aqueous solutions of zinc sulfate.
U.S. Pat. 935109, Sept. 28, 1909. British Pat. 605, Jan. 9,, 1909.

For the electrolysis of zinc sulphate solutions, or solutions of other sulphates, the anode consists of masure artificially prepared manganese peroxide free from constituents susceptible to the action of the electrolyte, and coated with ceresin. The anodes are prepared from a mixture of pure manganese nitrate and a binding agent, the mixture being pressed into moulds and heated.

ISHERWOOD, P. C. C., Leytonstone.

Treatment of refractory zinc-lead ores.
British Pat. 21584, Sept. 21, 1909. French Pat. 420510, Sept. 17, 1910.
German Pat. 238890, Apr. 2, 1910.
Abst., J. Soc. Chem. Ind., 1910, vol. 29, p. 1391.

SIEMENS and HALSKE, A.-G., Berlin.

Process for the electrolytic production of zinc from zinc pyrite cinders or other low-grade zinc-bearing raw materials, in which the raw material is leached out after a chloridizing roast.
German Pat. 233252, class 40c, from Nov. 17, 1909.
Abst., Z. angew. Chemie, 1911, vol. 24, p. 814.

STOGER, WALTHER.

Electrolytic zinc production.
Oesterr. Zeitschrift f. Berg- u. Hüttenwesen, 1909, v. 57, no. 1, p. 1-5. Ills.

The author describes in detail a process which is suitable even for low grade ores. A very pure product is obtained and the yield is between 90 and 97% of the original zinc content. The ore is roasted at a low temperature, the product extracted with dilute H2SO4, the solution carefully freed from metals of the Cu and As groups, and the zinc deposited by the electric current using Zn cathodes and Pb foil anodes. From a bath of 8 cathodes, and a current of 1500 amperes and 4 volts, 41.5 kg. of zinc are obtained in one day.

1910

ENGELHARDT, V. and HUTH, M.

Electrolytic extration of zinc.
Mettallurgie 1910, v. 7, p. 1-5.

Since the purity of the electrolyte is one of the essential conditions necessary for the electro-deposition of zinc, the use of a soluble anode, as in copper refining, is inapplicable, owing to the impurities of the anode contaminating the bath. The chloride and sulphate also

are the only salts economically available. Using the former of these, the Hoepfner process has been in successful operation for a year in spite of the difficulty of keeping the atmosphere free from chlorine. The greatest difficulty in the economical application of this process is the profitable disposal of the bleaching powder produced. The electrolysis of the sulphate appears to be more likely to be successful with ores low in zinc content, the sulfuric acid produced at the same time being used again for the extraction of more ore. The causes of a spongy deposit, and the condition necessary for the formation of a coherent metal are discussed. With the use of lead peroxide (PbO_2) at the anode, a coherent electrolytic zinc has been obtained, containing 99.98% of zinc, with the consumption of 3.9 kilowatt-hours per kilo (2.2 lb. of zinc), and the production of sulfuric acid of 10% concentration. The authors cal attention to the advantages and successful use of manganese peroxide as the anode.

FREUNDLICH, H. and NOVIKOW, W.

Electrolytic formation of zinc leaf on surfaces.
Z. Elektrochem. 1910. v. 16, p. 394-400.

The authorities have attempted to throw further light on the phenomenon investigated by Mylius and Fromm (Med. Ann. 1894, v. 51, p. 593.) If commercial zinc sulphate is electrolyzed in a flat dish with a platinum point just at the surface of the liquid as the cathode, the Zn separates as a thin lustrous film round the cathode. Mylius and From discovered that O was necessary for the occurrence of the phenomenon and the present authors have further established that there must also be on the surface of the electrolyte a film of some insoluble liquid. The necessary presence of oxygen is not accounted for by oxidation of the zinc, but by its favorable influence on the preliminary separation of amorphous zinc sponge. Many subst ances (ammonium salts, tartaric and citric acids, potassium cyanide) prevent the separation of zinc as a surface film, probably because they accelerate the transformation of zinc sponge into the crystalline variety. The separation of zinc sponge, and the binding of this into a surface film by the necessary layer of non-miscible material seem to be the only phenomena involved. The binding material may be an insoluble solid material so long as it gives the necessary thickness of 0.5 to 2 micro-millimeters.

KERN, E. F., Knoxville, (Tenn.), Assignor to P. S. Brown, N.Y.

Electrolyte and method of electro-depositing zinc.
U.S. Pat. 967200, Aug. 16, 1910.

The electrolyte consists of a solution of zinc fluosilicate, a fluosilicate of metal standing higher than zinc in the solution pressure series, such as the aluminium salt, an organic addition agent and ammonium fluoride. The last salt may be added periodically in portions during the electrolysis and, as an organic agent, grape sugar (dextrose) may be employed.

SANNA, E.

Electrolytic process for low-grade zinc ores.
Engng. Mining Jour., 1910, v. 89, p. 1106.

The author refers briefly to the various methods devised for obtaining Zn by means of the electric current and recommends his own process which is of special advantage for the treatment of low grade zinc ores and low-grade dumps. The zinc mineral is dissolved by alkaline salts, followed by electrolytic separation. The Zn is obtained as a metallic sponge which is then shaped in compact blocks by means of pressure and heat combined. The author estimates the cost of treatment at $75 per ton of Zn produced.

1911

DEKKER, N. H. M., Paris.

Treatment of ores.
British Pat. 18488, Aug. 16, 1911. French Pat. 445768, July 4, 1912.
Abst., J. Soc. Chem. Ind., 1912, vol. 31, p. 930.
Chem. Techn. Repert. 1913, vol. 37, p. 87.

HUTH, M., Assignor to Siemens and HAISKE A.-G., Berlin.

Manufacture of manganese peroxide anodes.
British Pat. 15128, June 28, 1911. (Addition to Br. P. 605-1909). U.S. Pat. 1043537, Nov. 12, 1912.

The anodes of MnO_2, manufactured as described in the main patent, are made more electrically conductive by embedding conductors in the form of rods or pieces of PbO_2 in the manganese nitrate, or by mixing manganese nitrate, and crystalline manganese peroxide.

KERN, E. F., Assignor to P. S. Brown, New York.

Electrolyte for zinc, and method of depositing zinc.
U.S. Pats. 999655, Aug. 1, 1911. German Pat. 244432, Jan. 26, 1911.

The electrolyte is a solution of $ZnCl_2$ containing Al^2Cl6 or another metal, with or without an alkali chloride. It may also contain grape sugar or some other organic addition agent.

(2) The electrolyte is a solution containing $ZnC12$ and a double chloride of A1 and another metal which may be one of the alkalis. The alkali chloride may also be in excess.

LARSON, J. P. A. and HELME, G. K. L.

Process for the recovery of zinc, cobalt, etc., from the end-lyes of copper extraction.
Swedish Pat. 34407, July 24, 1911. German Pat. 80525, Nov. 24, 1912.

The end-lye is treated with an electric current of about 4 to 6 v. in order to decompose the Zn and Co salts, the strength of current being between 2 and 10 am. per sq. m. An insoluble anode is employed, but no diaphragm is placed between the anode and the cathode. The Zn, Co, and Cu deposit on the cathode, while the acid or the Cl, not bound by the base, is neutralized with metals, metal oxids or metal hydroxides, etc. The solution is finally diluted in order to precipitate the Fe compounds.

MARINO, P.

Method of reducing soluble metal salts for the purpose of subsequently electrolytically depositing the metals present in said salts.
British Pat. 24562, Nov. 4, 1911. French Pat. 449822, Oct. 24, 1912.
Abst., Chem. Ztg. Rep. 1913, v. 37, p. 267.

The electrolyte, for use in the deposition of a metal or metallic alloy, is prepared from an aqueous solution of the salt or salts of the metal or metals to be deposited, to which is added an organic reducing agent, such as magnesium boro-citrate. The mixture is heated and, when cold, an organic reducing agent, such as gluco-citric or gluco-tantaric acid, is added to complete the dissociation of the salts in the solution.

MATUCHEK, J.

Electrolytic deposition of zinc.
German Pat. 244930, Apr. 22, 1911.

An aqueous solution of Zn fluoride, alone or with other zinc salts such as the sulphate, and preferably with addition of NH_4Fe, is used as electrolyte. Special claim is made for an electrolyte composed of 500 cc. of a solution containing 873 grms. of zinc sulphate.

NAMEKAWA, K., MIJAZAWA, K. and EMURA, S., Tokyo, (Japan).

Electrolytic refining of zinc.
U.S. Pat. 998379, July 18, 1911. German Pat. 264071, Mar. 15, 1911. French Pat. 430167, Mar. 16, 1911.

The zinc is deposited from a zinc sulphate solution on a suitable cathode and in order to increase the conductivity a current is passed from the solution to the lead anode at intervals, by which means the lead sulphate is converted into lead oxide which can easily be removed from the surface of the anode. The zinc is obtained as a coherent deposit by adding to the electrolyte a compound of vegetable origin which has been heated almost to the point of carbonization, and thick deposits are obtained by using a number of cathodes and lead anodes in a similar manner.

PAULIN, J. G.

Electrolysis of electrolytes in movement.
French Pat. 441316, Mar. 13, 1912. British Pat. 6261, Mar. 13, 1912. German Pat. 259151, Apr. 7, 1911.

The anode consists of a thick perforated plate of lead, and the cathode, of a thick perforated plate of zinc, if zinc sulphate is to be electrolyzed. The electrolyte enters through a tube and traverses a porous diaphragm, the sulfuric acid formed leaving through another tube. In order to regulate the speed of the electrolyte and keep it as nearly as possible uniform, a small sample of the liquid, near the upper end of the lower diaphragm, is led off through a tube to an electrolytic cell containing suitable electrodes. If the amount o- the sulphuric acid in the escaping electrolyte rises above a certain percentage, a relay inf terrupts the current in a coil, and a valve is opened, thus admitting more electrolyte through the tube. The speed of the electrolyte is thus increased until the electromotive force of the cell falls to the normal amount, when the throttle-valve is again partially closed by the passing of the current in the coil. This method of regulation may be employed for the electrolysis of Na Cl, Na_2SO_4, $ZnSO_4$, etc.

SIEMENS & HALSKE, A.-G., Berlin.

Manufacture of manganese peroxide anodes.
British Pat. 15128, June 28, 1911. (Addition to British Pat. 605, Jan. 9, 1909.)
Abst., J. Soc. Chem. Ind., 1911, vol. 30, p. 1168.

TAINTON, U. C. and PRING, J. N.

Electrodeposition and refining of zinc.
British Pat. 7235, Mar. 23, 1911. U.S. Pat. 1059233, Apr. 15, 1913. French Pat. 441476, Mar. 18, 1912.

The metal is deposited from a solution of a zinc salt (sulphate) containing from 10 to 30% of (sulphuric) acid and from 0.1 to 1.0% of a celloid agent (gum tragacanth). A high current density (up to 500 amperes per sq. foot of cathode surface) is employed and it is stated that the metallic deposits produced are smooth, lustrous and of fine quality. The electrolyte, primarily intended for the deposition of zinc upon metallic surfaces, may be employed in the refining of zinc and also for secondary batteries or accumulators in which a zinc cathode is used. Claim is also made for the process of obtaining zinc by treatment of the latter is acid and deposition of metal from the resulting solutions by the above means.

WHITFIELD, R. C. V.

Electrolytic extraction of zinc.
Iron and Coal Trades Rev., 1911, v. 82, p. 707.
Reprinted in: Min. & Engng. World, 1911, v. 34, p. 1128.

Theoretically, the electrolytic extraction of Zn is a comparatively simple matter, but technical difficulties are so great, and the operative cost is so high that results have not been in any way satisfactory. With the market price of zinc approximately 28 per ton it follows that there are fixed and definite limitations to the cost of production, and that if ores and zinc-bearing materials are to be treated in a preliminary operation, such acids, etc., can be used as will increase the productive cost only to an extent sufficient to permit ample margin remaining to cover the cost of raw materials, electrolysis, melting of deposited metal, wages, depreciation, and all items of expenditure, while still leaving a reasonable surplus between productive cost and market value, as profit.

1912

BELASIO, R.
Electrolytic determination of zinc in presence of ammonium salts.
Ann. Lab. Chim. Centr. delle Gabelle, 1912, v. 6, pp. 239-44.

In presence of ammonium salts, none of the usual methods for the electrolytic determination of Zn is applicable; dark spongy deposits being obtained. Good results can, however, be obtained if the zinc be converted into oxalate and electrolyzed in presence of lactic acid and sodium of K lactate. The solution containing 0.1 to 0.3 grm. of zinc as sulfate, and which may also contain ammonium sulphate and ammonia, is rendered slightly acid, then diluted to 200 cc., and after dissolving 3 grms. of ammonium oxalate and 5 grms. of K2SO4 therein, the Zn is deposited by electrolyzing for 2-3 hours with a current of 0.4-0.5 ampere at 3-3.5 volts, then adding 2 cc. of lactic acid, and electrolyzing for about 1 hour with a current of 1 ampere.

BUDDEUS, W.
Process for obtaining zinc liquors free from iron from liquors containing FeO.
German Pat. 273064, Dec. 12, 1912.

Such liquors, as, e.g. the waste liquors from the Cu extraction of ores and cinder containing Cu and Zn, from which the Cu has been separated by Fe, are precipitated with so much base or carbonate that all the Fe4 and ZnO is separated. The filtered and washed slime, which contains CaSO4, if precipitated with lime, is dried with admission of air. During the drying, all the Fe"O is oxidized to Fe2O3. Fromt he product which consists now of Fe2 O3, ZnO and CaSO4, the Zn can be removed by acids in the form of its salts, since it dissolves before the Fe4O3. The amount of acid required by the Zn need be added, and the Zn-salt removed by pressing and washing. The Fe-free Zn liquors can be worked up into Zn products or metallic Zn, by the known processes.

ENGEZHARDT, V.
Recent advances in the electrolysis of aqueous zinc solutions.
Metall. u. Erz., 192-3, v. 1, 1912, p. 60-72.

FRENCH, A. G.
Process for treating complex zinc-lead ores containing manganese.
French Pat. 449569, Oct. 18, 1912. British Pat. 22078, Sept. 28, 1912. U.S. Pat. 1041060, Oct. 15, 1912.

The ore is roasted and then mixed with from 5 to 15% (on the original ore) of pulverized sodium bisulphate. The mixture is heated to dull redness in a reverberatory furnace, the temperature being kept below the sintering point in order to retain the mass in a loose powdery form, and then, after cooling, soluble sulphates are recovered by lixiviation with water containing a small proportion of sodium bisulphate. The process is intended specially for the separation of zinc and manganese from complex ores.

FREUNDLICH, H. and FISCHER, J.
Influence of colloids on the electrolytic deposition of lead.
Z. Blektrochem, 1912, v. 18, p. 885-91.

The effect of a number of colloids (gelatine, agar-agar, casein, dextrin, gum tragacanth) on the electrolysis of lead silicofluoride and perchlorate has been studied by the authors. These two salts are the ones which under normal conditions give use to the most satisfactory deposits of the metal. In each case conditions were chosen where the deposit was decided by loose and then the minimum concentration of the colloid which produced a definitely smooth deposit was determined. These concentrations were compared with the protective action of the substances upon colloidal gold. The following figures show that for the author's experimental conditions there is no parallel between the behaviour of the colloid in those two cases:

Colloid	Minimum concentration in grams per litre to give a good deposit with:		Gold number in mgrms. per litre
	Lead silicofluoride 10.8 amp.	Lead perchlorate 2 amp.	
Agar-agar	0.3		0.6
Gelatin	0.6	0.2	0.15
Dextrin	1.0	0.3	1200
Gum Tragacanth	1.2	0.4	300
Casein	1.2		0.6

The general conclusion, however, is that in electrolysis the colloid is absorbed by the metal, causing a reduction in the velocity of crystallization, which in turn leads to a dense compact deposit. This assumption is supported by the observation that a similar influence on the form of separation is observed when lead is deposited from its salts by means of metallic zinc.

LARSON, J. P. A. and LAUR, G. K.

Electrodeposition of zinc, cobalt and other values from the waste liquors of Cu extraction, while at the same time rendering the liquors harmless.
German Pat. 280525, Nov. 24, 1912.

ISHERWOOD, P. C. C., Bushey Heath (Herts.)

Treatment of refractory zinc-lead ores.
British Pat. 22855, Oct. 7, 1912.
Abst., J. Soc. Chem. Ind., 1913, vol. 32, p. 1073.
Chem. Techn. Repert. 1914, vol. 38, p. 184.

PATERSON, J. H.

Investigation of the chemical reactions taking place at the cathode and anode during the electrolysis of simple salt solutions.
Proc. Univ. Durham Phil. Soc., 1912, vol. 4, pp. 187-220.
Abst., J. Chem. Soc. (London), 1913, vol. 104, pt. II, p. 100.
Chem. Ztg., 1912, vol. 36, p. 595.

SPEAR, E. B. and STRAHAN, S. G.

Determination of zinc by electro-analysis.
J. Ind. Eng. Chem., 1912, v. 4, pp. 889-91.

The authors find that the method proposed by Kemmerer for the electrolytic determination of Zn in alkaline solution is very satisfactory when slightly modified. The chief modifications consist in (1) cooling the solution to below 25° C. with ice if necessary, 5-6 minutes before removing the electrodes; and (2) using a smaller quantity of KOH, viz; 10-25 grms. for 125 cc. of solution. It is best to use a rotating nickel gauze cathode, completely immersed in the electrolyte, and a stationary platinum plate or spiral anode placed above the cathode. The solution should be free from nitrates and ammonium salts.

1913

ANONYMOUS,

Recent progress in the electrolytic zinc processes.
Revue Industrielle, 1913, v. 44, p. 720.

ARAKI, S., Osaka, (Japan).

Process of preparation of zinc sulphate from silicious zinc ores.
French Pat. 464038, Sept. 2, 1913. British Pat. 15546, July 5, 1913.
German Pat. 27309, Aug. 28, 1913.
Abst., J. Soc. Chem. Ind., 1914, vol. 33, p. 484.

BORCHGREWINK, H. K. and MOLSTAD, R.

Device for effecting circulation in electrolytes for the electrolytic separation of zinc.
German Pat. 277359, Feb. 20, 1913.

CHEM. FABRIK GRIESHEIM—ELEKTRON.

Process of rendering electrolytic zinc sponge permanent.
German Pat. 282234, June 29, 1913.

The process consists in removing adhering electrolyte and drying with exclusion of air. The electrolyte is removed by washing with H_2O after removal from the bath. If the solution is strongly alkaline, it is neutralized by passing CO_2 through it, since soda is more readily washed out than alkali. A low temperature, with stirring, and vacuum are employed preferably in drying the sponge, and dehydrating agents may be added. In this manner a zinc sponge of 90-95% Zn content is obtained, which is permanent in the air and exceeds zinc dust in activity. Furthermore, on account of its purity and uniformity of structure and properties it is more suitable than Zn dust for various reactions.

FRENCH, A. G., Nelson (British Columbia).

Electrolytic recovery of zinc and manganese.
U.S. Pats. 1055157 and 1055158, Mar. 4, 1913.

For the recovery of pure metallic zinc and manganese dioxide, and electric current of from 2½ to 4½ volts is passed from a lead anode to a zinc cathode immersed in an aqueous solution of zinc sulphate, manganese sulphate and normal sodium sulphate. The electrolyte after being acted upon is continuously delivered to the leaching tank from which the original supply was obtained, and is continuously replaced in the electrolytic tank by a supply drawn from the same leaching tank. (2) The electrolyte after being acted upon, and before delivery to the leaching tank, is treated with SO_2 or a solution of sulphurous acid in order to convert the MnO_2 suspended therein into manganese sulphate.

LIEBIG, MAX R. G.
Zink und Cadmium.
Thre Gewinnung aus Erzen und Nebeuprodukten. 205 fig. 10 plates.
Leipzig, Verlag von Otto Spamer, pp. 566-80.
The production of zinc by electrolytic methods.

MARC, R.
Influence of addition of colloids on the cathodic and anodic processes, in the electrolysis of metallic salt solution (lead and zinc).
Z. f. Elektrochemie, 1913, v. 19, pp. 431-44.
The author has examined the effect of additions of gum arabic, gelatine, gum tragacanth, dextrin, starch and albumin on the electrolytic deposition of zinc and lead from different electrolytes. The results are described in detail and tabulated, but cannot be generalized, for whilst colloids act favourably by diminishing the size of grain of the deposit, they frequently have an injurious effect on the ocherence and strength of the deposit. The favourable effect of clove oil on the character of electrolytically deposited lead is confirmed. Clove oil is non-colloided in character; it causes a diminution in the size of grain of the deposit, but apparently has no other effect, In general, colloids appeared to cause an increased liberation of H at the cathode, but their effect on anodic reactions varied according to the nature of the electrolyte and of the colloid.

Present status and future prospects of the metallurgy of zinc.
A symposium before the New York Sections of the Amer. Electrochemical Society and the American Inst. of Mining Engrs.

RICHARDS, J. W.
Electrolytic zinc.
Met. Chem. Engng., 1913, Dec., vol. 11, No. 12, p. 678.
Metal u. Erz. 1912-13, vol. 10, p. 985.

SIEMENS & HALSKE AKT. GES. and HUTH, M.
Anodes of massive MnO2.
German Pat. 279192, Nov. 9, 1913. British Pat. 27365, Nov. 27, 1913.
(Patent of addition to British Pat. 605-1909.)
These are made by carrying out the principal process in stages, to obtain increased density and conductivity. In one method, bodies of the dioxide, obtained by heating Mn material in molds, are allowed to cool, and are then reheated with more raw material, which is absorbed. In another method, (MnNO3)2 is added continuously during the heating and decomposition of the material already in the molds.

DURANT, H. T. and METALS EXTRACTION CORPOR.
Zinc sulphate.—Leaching of roasted zinc sulfide ores.
British Pat. 21737, Oct. 29, 1914.
Illus. Official Jour. Patents, 1916, No. 1414, p. 4733.

1914

INGALLS, W. R.
The metallurgy of zinc in 1913. (Hydrometallurgical processes)
Engng. and Mining Jour., 1914, vol. 97, p. 104.

LORDIER,
Zinc electrometallurgy.
Revue Scientifique, Industrielle et Commerciale des Métaux et Alliages, 1914, v. 7, No. 9 p. 2-4; No. 12, pp. 1-5.
C. V. Lordier reviews at length the electrometallurgy of zinc by wet (in No. 9) and dry methods (in No. 12). The zinc dress obtained in the Roessler Edelmann process of treating argentiferous ores contain about 70% zinc, and may be electrolytically refined with advantage. The finely granulated dross, spread on lead plates, forms the anodes, whilst rotating discs of zinc form the cathodes. The essentials of the Luckow (extraction of Zn from its minerals), Blast & Nuêt, Létrange (indirect method), Siemens & Halske, Cowper-Coles, Nahusen, Burghardt, Hoepfner, Dieffenbach processes are given. In conclusion it is stated that the only commercially successful processes in the electrometallurgy by indirect *wet* methods are these of Nahusen, Hoepfner and Dieffenbach. Electrolytes of ZnCl2 solution are said to be more economical in electric current than solutions of ZnSO4, the electromotive force necessary for the decomposition of the former being less than for the latter, although it is claimed by Gabran, that the electrolyte which gives the best results in the rapid deposition of zinc is a zinc sulphate solution of the following composition:

	Parts
Zinc sulphate	1200
Water	6000
Sulphuric acid (24° Bé)	60

MAZZUECHELLI, ARRIGO, Univ. Rome.
The cathodic deposition of metals in the presence of organic bases. 1. Zinc.
Atti accad. Lincei, 1914, v. 23-11, pp. 503-08.
Mazzuechelli recently observed that quinine has the same effect as gelatine and other colloids on the cathodic deposition of 3n and now reports the same effect with other organic

bases. The electrolytic precipitation of Zn from portions of solutions of 150 grm. crystallized $ZnSO4+50gm.$ $(NH4)2SO4$ in 1l. $H2O$ was studied. An As-free Zn-rod was the anode and a brass plate the cathode. Quinine, cinchonine, quinoline, dimethylaniline, α-naphthylamine, lutidine, aniline and pyridine were the bases added. In general, when a small amount of the base was added, in all cases during the first moments of electrolysis, a uniform, homogenous, fine-grained deposit of microscopic crystals was formed. On continuing the electrolysis or with larger c. The Zn is separated unevenly in cryst. warts. In the case of gelatine the first effect is more pronounced. The mechanism of the process in general is this: the base of its action is absorbed by the metallic granules that are first precipitated and keeps the granules fine according to the theory of Muller and Babmtje. On continuing the electrolysis the organic compound covers the cathode like a varnish and makes it passive except at a few thinly covered spots. At these the deposition continues and produces the warty deposits. It depends on the amount of the base absorbed in the beginning, on its capacity for redissolving in part and on its greater or less impermeability whether the favourable influence exercised at first is changed to a harmful one. Since no similar retardation of the rate of solution of the Zn anode takes place, the electrolyte solution of the anode is an irreversible process in the presence of organic bases, a solution containing such a base when electrolyzed between 2 Zn electrodes will give a continuous stream of H2 with an a.c. (42 cycle). Measurements showed that this evolution of H proceeds at an overvoltage of 0.130-0.180 v. The author intends to extend the experiments to various bases and metals.

II. Ibid.—626-33. Mazzuechelli here reports quantitative data. A careful review is given. M. has measured the voltage during the passage of a current of known density and also the voltage when the solution is at rest, using a Zn-coated brass cathode and a solid Zn anode in the $ZnSO4+(NH4)2SO4$ solution. The experiments were made at 11 in contact with the air, without a diaphragm, but with constant stirring. The electrical potential was measured by the usual compensation method, using a weston cell and measuring the v. between the electrode and a 9% Zn-amalgam in an equivalent N-$ZnSO4$ solution The $ZnSO4$-solution was electrolyzed without additions, with 5 x 10-5 parts of quinoline with 1 x 10-4 gelatine under various current densities and at a maximum voltage of 0.022. The potentials of the anode and cathode were measured before passing the current, during the passage of the known current. The individual potentials are higher during electrolysis than when the solution is at rest. With gelatin the excess potential is comparatively small which is in accordance with the fact that it shows less tendency to give a warty deposit.

PRING, J. N. and TAINTON, V. C.

Electro-deposition of zinc at high current densities
Chem. Soc. Trans. (London), 1914, v. 105, pp. 710-24.
Abst., J. Soc. Chem. Ind., 1914, v. 33, p. 423.

At current densities up to 40 amperes per sq. dom. the satisfactory electro-deposition of zinc can be effected in the presence of free acid up to 15 grms. $H2SO4$ per 100 cc. At which concentration the efficiency is highest. The potential differences are about 5 volts with lead and 3 with zinc anodes. The presence of colloids causes the production of smooth bright-surfaced deposits and permits the use of higher current densities.

RICHARDS, J. W.

Electrolytic deposition of zinc.
Trans. Amer. Electrochem. Soc. 1914, v. 25, pp. 281-90.

The refining of impure zinc in the absence or presence of precious metals may be commercially successful with cheap sources of power and high market values of zinc. Processes for the direct estimation of Zn in compounds, using soluble anodes containing Zn-compounds or no zinc, or using insoluble anodes, but utilizing or not utilizing the electro-chemical reaction at the anode, have either been abandoned or only moderately successful commercially, with the exception of direct estimation using insoluble anodes in which the electro-chemical reaction at the anode is not utilized. Ordinarily, zinc can only be efficiently deposited from slightly acid solutions with a low current density.

WATTS, O. P. and SHAPE, A. C.

Addition agents in the electro-deposition of Zn from zinc sulphate solutions.
Trans. Amer. Electrochem. Soc., 1914, v. 25, p. 291-96.

The most promising addition agents for obtaining good electrolytic deposits of zinc were found to be likonogen, pyrogallol and B-naphthol, the latter being the best, though gum arabic, benzoic acid, caffeine, formin, and terpinhydrate gave fair results, the most common defect being the production of vertical grooves or striations: One grain of the agent per litre was added to a solution of zinc sulphate (25% $ZnSO4.7\ H2O$), per square foot.

WILSON, F. L. and BRETHERTON, W. E.

Process for the precipitation of zinc.
U.S. Pat. 1083785, Jan. 6, 1914.
Abst., J. Soc. Chem. Ind., 1914, v. 33, p. 143.

1915

ANONYMOUS.

Anaconda to build big zinc reduction plant. (Special correspondence.)
Min. Engng. World, 1915, Dec. 25, vol. 43, pp. 1013-1014.

ANONYMOUS.
Kinematographs of electrolysis.
Elektrochem. Z. 1915, v. 21, p. 243-7.
Parts of films are shown. These were taken in the electrolysis of Hg, Zn, Pb and Sn.

FOERSTER, DR. FR.
Elektrochemie wasserigen Losungen.
(vol. I, of: Handbush der angew. physikalischen Chemie, in Enizeldarstellurgen herausgegeben von Dr. Georg Bredig.) 2d edition, Leipzig, Verlag von Johann Ambrosium Barth, 1915

HALL, C. A., Mount Airy, (Pa).
Electrolysis of zinc solutions.
U.S. Pat. 1163911, Dec. 14, 1915.
Abst., Met. Chem. Engng., v. 14, No. 3, p. 165.

ESHERWOOD, P. C. C.
Dense anodes of MnO2 for electrolysis of Cu, Zn or Mi from solution.
U.S. Pat. 1143828, June 22, 1915.
These anodes are made from Mn ore by leaching it with HNO3 or H2SO4, to remove Fe Ni and Co from the ore, forming a pasty mixture of the leached ore with a solution of Nn (NO3)2, molding, drying and heating sufficiently to decompose the nitrate and repeatedly soaking in Mn(NO3)2 solution, and drying and heating until the pores of the material are compactly filled.

PERREUR-LLOYD, M., Boulogne, (France).
Apparatus for the electrolytic production of zinc.
British Pat. 4681, Mar. 25, 1915. Same as French Pat. 472764, of 1914.
Abst., J. Soc. Chem. Ind., v. 34, 1915, p. 497.

PETERS, DR. FR.
Improvements in the electrometallurgy of zinc.
Electrolysis of aqueous solutions.
Glückauf, 1915, v. 51, No. 25, pp. 610-14.
A review of patents and articles from periodical literature.

SCHNABEL,
Handbuch der Metallhüttenkunde, vol. II, p. 114, p. 210, 116, 169, 214-15.

VADNER, C. S., Salt Lake City, (Utah).
Process of recovering zinc from an acid sulphite solution.
U.S. Pat. 1163286, Dec. 7, 1915.
Abst., J. Soc. Chem. Ind., v. 35, 1916, No. 2, p. 124.

1916

ANONYMOUS.
The Anaconda betterments.
Engng. Min. Jour., 1916, Feb. 19, v. 101, pp. 361-362.
Particulars of new plant and developments.

ANONYOMUS.
Anaconda Copper Co.'s zinc refinery construction.
Mining Eng., World, 1916, June 17, vol. 44, pp. 1127.

ANONYMOUS,
The bisulphite process.
Engng. and Mining Jour., 1916, Nov. 18, vol. 102, pp. 895-898.
Zinc sulphide is roasted to an oxide; changed to a bisulphite with sulphuric and leached from the foreign material. Flow sheets of the original and modified bisulphite processes are given.

ANONYMOUS,
Electrolytic zinc production.
Engng. Mining Jour., 1916, Jan. 1, v. 101, pp. 25-26.

ANONYMOUS,
The French electrolytic process.
Min. Engng. World, 1916, Sept. 9, vol. 45, No. 11, p. 450.

ANONYMOUS,
The French zinc process at the Standard Property, B.C.
Min. World, vol. 44, 1916, Apr. 15, No. 16, p. 741.

ANONYMOUS,
The French zinc process at the Standard Silver-Lead Mining Co. at Silverton, B.C.
Min. Engng. World, 1916, Apr. 15, vol. 44, p. 741.

ANONYMOUS,
Isherwood's zinc process.
Mining Mag., (London), Aug., 1916, v. 15, No. 2, p. 103.
Announcement of the Venture Trust regarding the formation of the British Electrolytic Zinc Company, with the object of working Dr. P. C. C. Isherwood's process for producing a pure electrolytic zinc from complex sulphide ores.

ANONYMOUS (Editor),
Mill and smelter construction in 1916.
Anaconda Copper Co.—Electrolytic zinc.
Work done at the Consolidated Mining and Smelting Co. of Canada, Trail.
Min. Engng. World, 1917, Jan. 6, v. 46, No. 1, pp. 2-7.

ANONYMOUS,
Progress in electrolytic zinc manufacture.
Min. and Engng. World, vol. 44, Apr. 1, 1916, No. 14, p. 655.

AUSTIN, L. C.
Electrolytic zinc.
Min. Sci. Press, 1916, v. 112, p. 434.

BRETHERTON, S. E.
Electrolytic practice.
Mining and Scientific Press, 1916, Dec. 2, v. 113, No. 23, pp. 793-794.
Correspondence speaking of previous penalizing for zinc in some ores and the doing away of this through the introduction of the electrolytic process developed by the Anaconda Co.

HALL, CLARENCE, Mount Airy (Pa.)
Method of zinc electrolysis.
U.S. Pat. 1163911.
Abst., Engng. and Min. Jour., v. 101, 1916, No. 6, p. 263.

HANSEN, C. A.
Electrolytic zinc at Bully Hill.
Met. Chem. Engng., 1916, v. 14, No. 3, pp. 120-22.
Min. Magazine, 1916, v. 14, pp. 176-77.

HANSEN, C. A. (General Electric Co.)
Electrolytic zinc at Bully Hill.
Met. Chem. Engng., v. 14, Feb. 1, 1916, No. 3, pp. 120-22.

INGALLS, W. R.
Comments and speculations on the metallurgy of zinc.
Engng. Min. Jour., 1916, Oct. 7, v. 102, No. 15, p. 621.

INGALLS, W. R.
Electrolytic zinc.
Trans. Amer. Electrochem. Soc., v. 29.
Engng. Mining Jour., 1916, v. 141, pp. 425-28.
Mining Mag., 1916, v. 14, p. 234.
Met. Chem. Engng., 1916, v. 14, pp. 264-5.
Description of Anaconda process, etc.

INGALLS, W. R.
Electrolytic zinc.
Engng. and Mining Jour., v. 101, Mar. 4, 1916, No. 10, pp. 425-28.
General discussion of conditions governing production of electrolytic zinc. Description of Anaconda process. Commercial and technical limitations. Quality of electrolytic spelter.

INGALLS, W. R.
Electrolytic zinc.
Min. and Sci. Press, v. 112, Mar. 25, 1916, No. 13, pp. 439-42.

INGALLS, W. R.
Electrolytic zinc.
Paper read at the meeting of the New York section of the Amer. Electro chemical Society on Feb. 11.
Met. Chem. Engng., vol. 14, 1916, No. 5, p. 264-5. (Abst.)

LAIST, F. and FRICK, F. F.
Extracting zinc from ores.
U.S. Pat. 1167700, Jan. 11, 1916.
Abst., Chem. Abst., v. 10, 1916, No. 6, p. 745.

LAIST, FR. and FRICK, F. F., Anaconda.
Hydrometallurgy of zinc with electrolytic deposition.
U.S. Pats. 1167700-701, Jan. 11, 1916.
Abst., Met. Chem. Engng., v. 14, 1916, No. 4, p. 220.

LAIST, F. and FRICK, F. F.
Process for extracting zinc from ores.
U.S. Pat. 1167701 Jan. 11, 1916.
Abst., Chem. Abst., v. 10, 1916, No. 6, p. 745.

LYON, D. A.
Hydrometallurgy of zinc and lead in 1915,
Met. Chem. Engng., 1916, Jan. 1, v. 14, pp. 30-32.
(New Anaconda plant of 25 tons metallic zinc per day per day nearing completion.)

MORGAN, H. J. and RALSTON, O. C.
Production of zinc dust by other methods than in the zinc retorts.
(Electrolytic zinc dust).
Met. Chem. Engng., 1916, Oct. 15, v. 15, No. 8, pp. 465-68.
Abst., Mining Mag., (London), 1916, v. 15, No. 6, p. 364.

MOTHERWELL, H. A. B.
Electrolytic zinc.
Min. Sci. Press, 1916, Mar. 18, v. 112, pp. 401-03.

MOTHERWELL, H. A. B.
Electrolytic zinc.
Mining and Sci. Press, vol. 112, Mar. 18, 1916, No. 12, pp. 401-03.

RICKARD, F. A.
Electrolytic refining at Trail.
Min. Sci. Press, 1916, Dec. 23, v. 113, No. 26, pp. 903-907.

RYAN, J. D.
Progress in electrolytic zinc manufature.
Min. Engng. World, 1916, April 1, v. 44, p. 655.
(Statement made by J. D. Ryan, President of the Anaconda Copper Co.)

STEVENS, R. H.
The Peugeot process of zinc electrolysis.
Salt Lake Mining Rev., 1916, July 30, v. 18, No. 8, pp. 17-19.
A complete but concise description of the method is given. Cost data are included

VADNER, CH. S., Salt Lake City.
Process of recovering zinc from an acid sulphite solution.
U.S. Pat. 1163286.
Abst., Min. Sci. Press, vol. 112, 1916, No. 14, p. 483.

GRUNNER, VILHELM,
Electrolytic precipitation.
Tidskrift Kem. Farm. Terapi, 13, 216-20, 232-6.

Zinc is electrolytically precipitated in compact form from pure acid solutions if sufficiently stirred. The zinc concentration may be as low as 2% and the acid as high as 30%. At a zinc concentration of 5% and acid concentration of 5% the maximum precipitation occurs when $Dk = 500$ amp. per sq.cm. Lowering the zinc concentration or increasing the acid concentration lowers the rate of precipitation. Increasing the acid concentration also lowers the electric tension. At about 10% the energy consumption per kilogram of zinc is at a minimum. The precipitation is somewhat facilitated by the addition of water glass, but the addition of hydro-fluosilicic acid gives no advantage. Electrolytic zinc heated to just under its melting point is increased in volume from 15% to 20%. In most concentration the Zn-ion concentration is increased by the addition of H_2SO_4, provided, however, that this is not at a point where complex salts are formed.

ANONYMOUS,

Zinc burning.

The *Metallbank und Metallurgische Ges. Akt.-Ges.* in German patent, 290,013, Oct. 8, 1913 (addition to 252,195) says that the driving off of the zinc from the mixture of ore and fuel has been found to proceed more satisfactorily the smaller the excess of air in the heating gases entering the reaction chamber. Upon smelting the charge a portion of the zinc is held back by the slag which forms, while the reduced zinc vapor is reoxidized by the excess air to ZnO, which is dissolved by the molten slag. In order to overcome this difficulty, the process is carried out in a reducing atmosphere, avoiding fusion. In order that the highest possible temperatures may be used, the charge is mixed with ores or additions so that only a sintering results, without fusion. The CO_2 present in the heating gases has been found sufficient to oxidize the zinc vapors without excess of air. The zinc vapours may be oxidized by air subsequently.

DISCUSSION

MR. R. H. STEWART: The Trail process differs somewhat from the Anaconda process as described by Mr. Mathewson. Following the practice at Great Falls the ore is leached until the free acid is reduced to two or three-tenths of one per cent; the electrolyte is then drawn off the still acid pulp and neutralized, fresh calcine being used for this purpose instead of limestone as at Great Falls. The calcine is then returned to the leaching tanks and is used for the treatment of spent electrolyte. In other respects the two processes are practically the same. The units composing the Trail plant are, however, smaller than those at Anaconda. The Trail plant when completed, will contain thirteen 1,000 k.w. units. Each unit consists of two 250-volts, 4000-ampere, direct-current generators, driven by one synchronous motor operating at 2300 volts. Each unit is connected up with thirty-two tanks in series. The current used is from 25 to 30 amperes per square foot of cathode surface. Purification of the solutions is accomplished in practically the same way as at Great Falls.

MR. J. C. GWILLIM: In travelling through British Columbia recently, I found that the zinc-ore produced there was being sold to different ore purchasing companies, for consignment to reduction works in the middle western States as well as to Anaconda, at Great Falls, and also to Trail. Apparently the Anaconda works had not the same objection as Trail to the presence of iron in the ore. The British Columbia zinc ore carries much iron and the miners go to a great deal of trouble to remove it. The representative of the Anaconda Company, however, purchased ore containing as high as 30% iron without penalizing it. I should therefore like to be informed whether in the treatment of such

ore at Anaconda, no trouble is experienced on account of the iron content.

MR. STEWART: At Trail the difficulty with the iron is in the roasters. The iron and zinc combine to form a compound which is not soluble in the solutions used for the leaching. Most of the ores we have treated have been difficult to roast and the iron has caused trouble in this manner.

MR. E. P. MATHEWSON: So far as the process used by the Anaconda Copper Mining Company is concerned the presence of iron in the concentrate is no serious detriment. The iron is removed as soon as the solution is neutralized with powdered limestone.

MR. E. E. WATTS: I would like to ask Mr. Mathewson what his experience has been in limiting the amount of impurities in his electrolyte.

MR. MATHEWSON: The electrolyte used at Anaconda and Great Falls was prepared in such a manner as to contain practically no impurities, for if any impurities were left in the electrolyte it would make serious trouble in the character of the deposited zinc.

A MEMBER: I would like to ask what percentage of the zinc present in the ore is recovered from the electrolyte and what percentage goes into the residuum?

MR. E. P. MATHEWSON: The percentage received varies greatly according to conditions, but it may be stated that there is no difficutly in obtaining 90% of the contents of the ore in the leaching solution.

MR. GWILLIM: The electrolytic zinc process is apparently suitable for treating complex zinc ores containing values in silver; the Standard or Belgian ovens process is not suitable for the treatment of such ore, nor does it recover the silver values as well as the electrolytic process. Hence the electrolytic process has a field of operation in treating complex zinc ores high in silver values. What I want to know is, could the electrolytic process successfully compete with the Belgian oven process working on purer zinc ores, such as those of Missouri?

MR. MATHEWSON: Such ores would be treated easily by the electrolytic process.

MR. H. J. ROAST: At the present time our zinc dross, a by-product of the galvanizing process containing 92% zinc, is all sent to the United States where the zinc is recovered. It seems very desirable to have this recovery undertaken in Canada. At the present time a company with which I am associated is conducting experiments on a commercial scale in this connection. The results have been very encouraging; for instance, the iron content has been reduced from 6% to 0.03%.

MR. MATHEWSON: I would like to make a few remarks in regard to the duplication of experiments by various research chemists working on the same problem. The amount of this duplication has been tremendous in this one problem of securing a pure zinc electrolyte. The same experiments have been tried time and again, simply because results were not published promptly or not published at all. There should be some system of co-operation among research chemists whereby results could be declared promptly, provided they do not interfere with the financial affairs of the respective Companies for which the chemists are working. The great war is a tremendous stimulus to research and the research chemists have not had time to publish results on account of the necessity of producing the metal as rapidly as possible; in fact, the Anaconda process had not been described except in a general way because of the pressure brought to bear on the Research Department to get the larger plant in operation with the least possible delay. There have been a few details of the process which the Anaconda research chemists have found it advisable to patent for the sake of protection, but really fundamental patents on electrolytic zinc production seem to be unobtainable at this late date. Mr. Watts has referred to many experiments that have been tried not only by people he has mentioned, but by the General Electric Company and also the Anaconda Copper Mining Company and a numbers of others. If results had been published by any one of these Companies the others would have been saved unnecessary delay and expense. The American Association for Testing Materials has drawn up specifications for electrolytic zinc and among these a very small amount of cadmium

is allowed in the electrolytic zinc. The Anaconda Company spent a great deal of time and money in devising ways and means of reducing the quantity of cadmium in their electrolytic zinc, so as to come within the limits set by the specifications referred to, and finally devised a scheme whereby they could control the amount of cadmium exactly and found there was no difficulty whatever in keeping within the limits specified. In the course of experimentation they devised a tube mill made of cast copper and filled with zinc balls. This did the work very well and was specified for the large plant at Great Falls. Machinery was ordered accordingly for that plant and copper castings to the value of over $200,000 were prepared when the research chemists discovered that the tube mills were unnecessary, as the same thing could be accomplished in a much similar manner, using zinc in powdered form, so these copper castings were re-melted and sold. This shows that the processes are becoming simplified rapidly; in fact, the Anaconda process as in use to-day bears very little resemblance to the process as used a year ago. Several producers of electrolytic zinc have succeeded in marketing a uniform product of the highest degree of purity. I would like to emphasize the fact that the most successful processes, in this, as in nearly everything else, are those which are the simplest.

ELECTROLYTIC ZINC IN EASTERN CANADA

By E. E. Watts

Annual Meeting, Montreal, 1917

Attention has recently been directed to the production of zinc from Canadian ores, and particularly to the production from the zinc ores of British Columbia.

Though the deposits of zinc ore in Eastern Canada are small as compared with Western deposits, sufficient zinc could possibly be produced to supply the present industrial requirements of the country.[1] Furthermore, under normal conditions, plants producing zinc in Eastern Canada would have the following advantages over plants in Western Canada: 1, Proximity to markets; 2, cheaper labour; and 3, accessibility to supplies and equipment.

At the present time there are two operating zinc mines in Eastern Canada. Both are in Montauban Township, Portneuf County, Quebec, and both are located on the same ore body. At one, the Zinc Company, Limited is daily treating about 125 tons of ore which assays about 10% zinc and 2% lead. The entire millfeed is ground in a ball-mill, classified, and tabled on Butchart tables and a Deister Slimer for the recovery of lead. Middling from the Butchart tables is retabled on James tables for the further recovery of lead, and some zinc-iron concentrate is saved. Zinc middling from the James tables is treated by Mineral Separation machines to recover a zinc-iron flotation concentrate. The zinc-iron concentrates are combined, are given a flash roast, and are treated on magnetic separators to obtain a 40% zinc shipping product.

At the Tetreault mine, on the adjoining lot, about 60 tons of ore is treated daily. The mill feed assays about 14% zinc and 3.5% lead. Joplin practice is followed and good results are being obtained by the use of jigs and tables. The crushing is done by rolls.

[1]Production of Spelter in Canada, 1916—Dept. of Mines.

The Canadian Zinc Products Company, which is under the same management as the Zinc Company has recently completed a zinc oxide plant, equipped with Wetherill grates, and also has had an experimental electric zinc furnace in operation at Shawinigan Falls. This electric furnace was designed by Dr. Alfred Stansfield for the manufacture of spelter and zinc dust, and interesting results have been obtained.

With the exception of the small quantity of ore treated by the above company and by the Electro Zinc Company of Welland, the ore mined in Montauban Township has been shipped to the United States for treatment.

Possible methods of treating zinc ore for the recovery of zinc may be roughly classified as follows: 1, Smelting; 2, chemical methods; 3, electrolytic methods; and 4, electric smelting.

Smelting includes the standard distillation method for high-grade ore. Distillation may be preceded by igneous concentration.

Chemical methods include various chemical processes for the production of zinc salts of zinc oxide, possibly followed by smelting for the recovery of zinc.

Electrolytic methods include processes for the recovery of zinc by electrolysis, in which electrolysis may be preceded by preparatory chemical or igneous processes.

Electric smelting includes methods for producing zinc, zinc oxide, or zinc dust, by the use of the electric furnace.

For the treatment of Canadian zinc ore in Canada for the production of zinc, the employment of electrolytic methods is now generally regarded as the most promising solution.

Experimental work.—In 1912 and 1913, at the School of Mining, Kingston, the writer experimented with zinc ore from the Sullivan mine at Kimberly, B.C. The ore was a dense complex sulphide of which the proximate chemical analysis was:

Zinc	31.2%	Insol	12.8%
Lead	10.1%	Lime	Trace
Iron	13.4%	Sulphur	31.0%
Silver	3.1 oz.		

Microscopic examination showed that the diameters of the mineral particles were as follows;

Blende.....0.20 mm. to 0.05 mm.
Galena.....0.20 mm. to 0.05 mm.
Pyrite.....0.10 mm. to 0.025 mm.

The microscopic examination precluded the possibility of a mechanical concentration,[1] and hence left igneous or hydrometallurgical process as possible means of treatment.

Preliminary roasting and lixiviation tests indicated that a sulphurous acid treatment of the roasted ore would give interesting results. The conditions were worked out for obtaining an 87% zinc extraction by a sulphurous acid leaching, and attention was directed to the recovery of zinc from the resulting solution of zinc bisulphite. The conditions were determined for the precipitation of zinc monosulphite from the zinc bisulphite solution.

The Zinc Commission[2] has pointed out that zinc smelting in Canada, even for high-grade ore, is a very doubtful commercial enterprise. It would therefore appear that a sulphite precipitate must be roasted to oxide and exported, or the zinc must be recovered from it in some other way then by distillation.

Tests were conducted to determine if it was possible to obtain metallic zinc by the electrolysis of solutions of zinc bisulphite. The deposition of sulphur and zinc sponge, and the evolution of hydrogen sulphide made this electrolysis impossible. Then, attempts were made to obtain metallic zinc by electrolysis of solutions of zinc sulphate to which zinc sulphite precipitate was periodically added. The additions were made to the bath and to muslin bags at the anodes, but the tests showed that in the presence of sulphurous acid or zinc sulphide, successful electrolysis was impossible. Tests were conducted in which the zinc sulphite precipitate was first roasted to oxide before charging at the anodes in a zinc sulphate electrolyte. Successful results

[1]Recent work, proves that a lower grade and more coarsely crystalline Sullivan ore can be concentrated by the Horwood Process.

[2]Report of the Commission on the Investigation of the Zinc Resources of British Columbia.

were obtained. The zinc deposited in a good plate, and the electrolyte was maintained by the addition of the roasted sulphite; hence to free the bath from particles of precipitate that might attach themselves to the deposit, to place the roasted precipitate at the point in the bath where the sulphuric acid is regenerated by the electrolysis, and to decrease, if possible, the decomposition voltage by allowing the roasted precipitate to act in part as anode, the crude zinc oxide obtained by roasting the zinc monosulphite precipitate was placed in bags at the anodes of the electrolytic cell.

From the foregoing experiments the following conclusions were deduced:

1. Successful electrolysis is impossible in the presence of sulphurous acid or zinc sulphide.

2. Successful electrolysis is possible with an electrolyte of zinc sulphate to which zinc oxide, prepared in any suitable manner, is charged to compartments surrounding the anodes of the cell.

3. Lead forms a satisfactory anode, and copper, zinc, and aluminum form satisfactory cathodes.

4. With current densities of from 2.5 amps. per sq. dm. to 4.5 amps. per sq. dm., a current efficiency of approximately 100% can be maintained.

5. Depending upon the current density, the voltage varies from 3.5 to 6 volts.

6. The extraction of zinc from zinc oxide varies from 95 to 100%.

The process as outlined was patented, and was developed for the treatment of high-grade zinc ore. The process has been referred to as the Watts Process.

The high-grade zinc product for charging to the electrolytic cells may be roasted zinc concentrate, impure or refined zinc oxide, or zinc dust. The advantages of such a treatment are:

1. The solution is kept small in bulk, and the solution and tanks serve at once for leaching the ore and for electrolysis.

2. The acid generated at the anode is immediately used up

at the point at which it is generated, and at the point where it is most concentrated and hence most active. Thorough extraction is thereby obtained.

3. The bath is automatically kept at constant composition and purity, by zinc going into solution with the relative exclusion of impurities.

4. The zinc content of the solution is kept constant, and high current efficiency is therefore maintained.

5. The conditions are fulfilled for obtaining a good cathode deposit.

6. The residues from electrolysis may be further treated for the recovery of other metals.

Larger Scale Experimental Work.—The writer became asso ciated with the Weedon Mining Co., and in March 1915, carried on experimental work in the Metallurgical Laboratory at McGill University. Various ores were treated, including ore from the company's mine at Notre Dame des Anges, Que., zinc oxide from Leadville, Colo., and ore from Mineral Point, Wis. The work served to develop a wooden electrolytic tank, asphalt-lined and antimony-lead anodes and aluminum or copper cathodes. The anode compartment consisted of a wood and canvas box which surrounded the lead anode.

In August 1915, work was started on an experimental plant at Welland, Ont. The equipment consisted of a 100 K.W. motor-generator set, a blower for air agitation of the anode solution, a vacuum pump for filtering, 28 electrolytic tanks 3′ by 3′ by 4′ deep, sump tanks, storage tanks, and filter tanks. Standard cast-iron centrifugal pumps were used for handling solutions. An Abbe ball mill was used for grinding.

Roasted Joplin concentrate assaying 60% zinc was the first ore treated at the plant. It was necessary to charge this ore to the electrolytic tanks in the form of a paste, and a small pebble mill was used for mixing the ore into a paste with zinc sulphate solution. From the mill it was fed by hand to the anode compartments at six-hour intervals. With oxide it was possible to use dry charging, and in both cases, air pipes in the bottom of the compartments kept the ore particles in suspension.

In the first trial run in the plant, an unsuccessful attempt was made to reverse the position of the compartments. Air agitation was used in the bottom of the tanks, and the compartments surrounded the cathodes instead of the anodes. Seven anodes were used per tank. It was found that the solutions would not transmit 800 to 1000 amperes without overheating. With two anodes per tank, with the compartments surrounding the anodes as in the original experimental tank, and with a current of only 250 amperes per tank corresponding to a current density of 12.5 amperes per square foot, good results were obtained. But, operating in this manner reduced the plant capacity about 70%. Several expedients were tried for keeping the solutions cool when employing higher currents, but, the heat resulting in part from the electrolytic resistance and in part from the chemical reaction at the anode, made higher currents impractical with this form of tank. Considerable difficulty was also experienced with the compartments, for the canvas used as a diaphragm material developed into a serious weakness and required constant attention and repairs. Promising preliminary tests were made upon other diaphragm materials, but this investigation was not completed. Considerable trouble and loss was caused by the tank construction, and great difficulty was experienced in obtaining an absolutely reliable and tight tank; in fact, it was necessary to replace the original tanks, and the new tanks were made with approximately the same cross section, but considerably shorter in order to accommodate two or three anodes. It was possible therefore to run two series of tanks in parallel and thereby to double the plant capacity. The melting of the deposits, which were stripped every two to six days, caused trouble for a time. An attempt was made to strip copper cathodes by dipping them into a bath of molten zinc. Loss by drossing, damage to the cathodes, and copper entering into the zinc, made this impractical. Melting was thereafter done in a gas-fired crucible tilting furnace. A flux of sal ammoniac was found to be advantageous.

In general, the laboratory experimental results were attained, but our operations did not permit of a sufficient margin for profit. The company did not see their way clear to further perfect

the details of the process for large scale operation, so the plant was closed in August 1916.

When producing zinc at the rate of 8000 pounds per month, or at the rate of 270 pounds per day, the cost of producing zinc was as high as 18 cents per pound plus the cost of the ore.

On the basis of a steady production of 60,000 pounds per month or 2000 pounds per day, the writer estimates that the cost of producing zinc electrolytically should be 4 cents per pound when power is available at 14 dollars per horse-power year. The costs would be distributed as follows:

Crushing, pumping, filtering and purifying	0.6	cents per lb.
Electrolysis, including power, labour and repairs	2.3	" "
Melting	0.8	" "
General	0.3	" "
Total operating cost	4.0	cents per lb.

In a large plant the above costs would be considerably reduced.

Thirty tons of 60% ore were received at the plant, as well as ten tons of oxide that was manufactured from the ore mined at Notre Dame des Anges, Que. About 30,000 pounds of zinc were produced which averaged 99.95% zinc.

Conclusion.—The work showed that it was possible to operate upon a commercial scale, but it showed that weakness existed in the diaphragm material, and that many of the mechanical details could be improved. The process possessed advantage over other processes with respect to extraction, current efficiency, and necessary purity of the electrolyte, for good results were obtained in solutions in which ordinary electrolysis is impossible. With a satisfactory diaphragm it seems probable that a higher acid electrolyte could be used with a correspondingly high current density, and it is probable that future development will be along this line.

In conclusion, I wish to acknowledge my indebtedness to Professor S. F. Kirkpatrick of the School of Mining, Kingston; Dr. Alfred Stansfield of McGill University, and Mr. L. D. Adams, Pres. and Gen. Mgr. of the Weedon Mining Company.

THE FORMATION OF ZINC FERRATE[1]

An Explanation of the Low Solubility of some Roasted Zinc Ores in Dilute Sulphuric Acid.

By E. H. Hamilton, G. Murray, and D. McIntosh

Early in the development of the electrolytic method of producing zinc from its ores it was noticed that a large amount of the zinc could be extracted by dilute sulphuric acid, in a few minutes, but that the remainder could only be removed by prolonged boiling. This insolubility was not due, apparently, to the particles of the zinc compounds being protected by the lead sulphate in the calcine, for on removing the lead salt by appropriate methods no increased solubility was obtained. It was noticed, further, that on treating the residue insoluble in cold dilute acid with boiling acid, the zinc and iron dissolved approximately in the ratio of one atom of the former to two of the latter. Finally, it was ascertained that a high temperature in roasting increased the amount of insoluble zinc, although the calcine produced might contain less sulphur as sulphide than one made at a lower temperature.

The insoluble zinc must be due to the iron, since higher grade zinc ores gave greater percentages of soluble zinc; and the lead sulphate we have shown to be innocuous, so that the complexes formed with iron must cause the low extraction.

The compounds of zinc and iron oxides have been examined by a number of investigators. Roscoe and Schorlemmer[2] state that Ebelmen obtained black octahedra of the composition Fe_2O_3 ZnO by strongly ingniting the oxides with boron trioxide. Hofman (Trans. A. I. M. E. 1905) and Wells (*Eng. and Mine Jour.*) (86,420) showed that combination took place between these two oxides, but no attention was paid to temperature or time. Ingall's "Metallurgy of Zinc" (pages 6 and 32) mentions the formation of zinc ferrite (zinc ferrate.)

---

[1]Printed by permission of the Consolidated Mining & Smelting Co.

[2](Page 1225). The compound is referred to as zinc ferrite, and the name ferrate is reserved for the salt of the hypothetical acid H_2FeO_4. It would seem better to speak of the compounds of ferric oxides as ferrates, and of ferrous oxide as ferrites.

The results of the present investigations corroborate the observations of Messrs. Hofman and Wells, and, in addition, go more fully into the conditions governing the combination.

The first part of the work consisted of experiments with pure zinc oxide and ferric oxide. Later, some roasting and leaching tests were made on various ores to find out how far the results obtained from the pure compounds applied to the minerals.

Ferric oxide was prepared by treating ferric chloride with ammonia and roasting the washed hydroxide for two hours at a red heat. The zinc oxide used was Baker's 'chemical pure'. A mixture was carefully made of material ground to 120 mesh to correspond to the formula $ZnO\ Fe_2O_3$. This was reground to bring about a very intimate mixture, and the tests were made at the various temperatures in a muffle furnace heated electrically. A thermo-couple, correct at the temperature of melting zinc, indicated the temperatures, and these were maintained constant by varying a resistance in series with the muffle.

After the heat treatment the material was again ground and leached with a hot ammonia and ammonium chloride solution and the loss in weight found. From these results the amouut of zinc oxide that had entered into combination was determined.

The tabulated results of Tests No. 1 to No. 7 are given in Table 1, and shown in the curves (Plate 1).

Table I.

Experiment No. 1	Temperature 1600° F.
1.15 hrs.	100%
3.5 "	100%
5.00 "	100%
5.3 "	100%
Experiment No. 2	Temperature 1500° F.
5 hr.	100%
8 "	100%
Experiment No. 3	Temperature 1400° F.
1 hr.	88.4%
3 "	85.5%
5 "	89.3%
7 "	97.1%
9 "	100.0%

Experiment No. 4.	Temperature 1300° F.
1.15 hrs.	67.6%
2.75 "	76.8%
5.25 "	82.5%
7.25 "	85.2%
Experiment No. 5	Temperature 1200° F.
1.5 hrs.	35.0%
3.0 "	73.6%
4.5 "	70.4%
6.0 "	70.0%
Experiment No. 6	Temperature 1200° F.
2.0 hrs.	76.2%
4.5 "	84.4%
6.0 "	71.2%

Experiment No. 7—At 1100° F. no combination took place.

A few experiments to determine the formula of the compound were made by heating various mixtures so that combination was complete. In all cases where the iron oxide was in excess of the necessary amount for the formula ZnO Fe_2O_3 none of the zinc went into solution. On the other hand, while the results obtained with the zinc oxide in excess did not agree as well as could be desired, it is certain that the compound does not contain more zinc than is shown by the formula ZnO Fe_2O_3.

TABLE II.

Ratio of ZnO to Fe_2O_3	Theoretical loss in weight if formula is ZnO Fe_2O_3	Actual loss.
1.5 : 1	14.4%	17.1%
2 : 1	25.3%	20.9%
3 : 1	40.3%	50.9%

From the curves it will be seen that the temperature has a great effect on the speed with which the reaction takes place. At 1600° F. the combination is complete at the end of one hour.

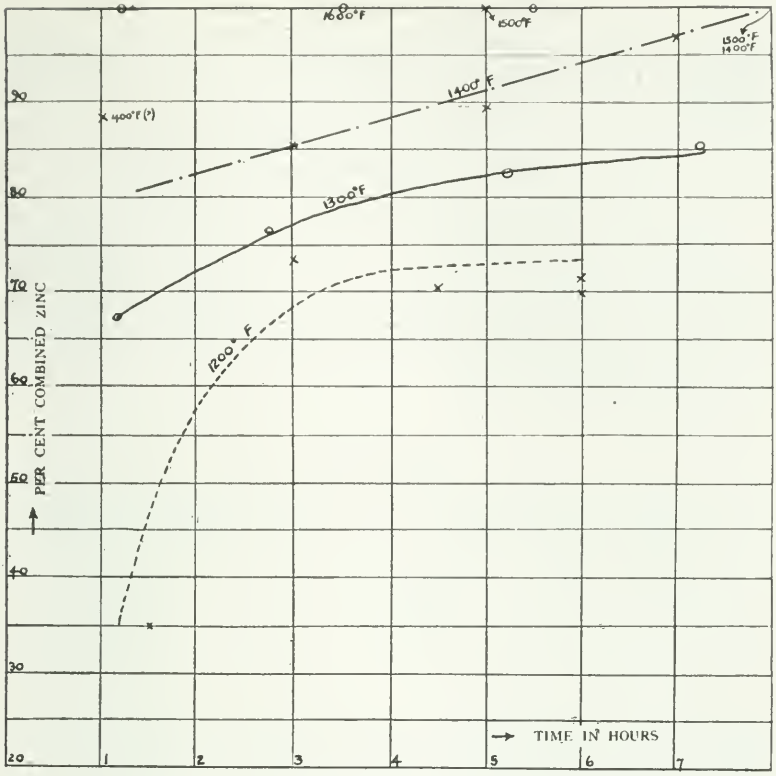


Plate 1.—Curves showing combination of zinc and iron at different temperatures.

At 1500° F. it is complete in 5 hours and probably sooner. At 1400° F. the reaction is 85%, complete in 3 hours and complete in 8 hours. At 1300° F. about 70% of the zinc has combined in 1 hour and 85% in 8 hours, while at 1200° F. it is 4 hours before the reaction is 70% completed. At 1100° F. no combination could be detected within 8 hours.

PROPERTIES OF ZINC FERRATE:

Zinc ferrate is a yellowish brown material, magnetic and never more basic than ZnO. Fe_2O_3. It is soluble in hydrochloric acid, insoluble in cold dilute H_2SO_4, H_2SO_3 and hot caustic solutions. It is, however, slowly soluble in hot dilute sulphuric

acid. We were unable to find any reagent which would dissolve the zinc without bringing a considerable quantity of iron into solution.

Combination of Zinc Oxide and Alumina:

During the investigation it was found that zinc oxide also combined with freshly prepared aluminum oxide made by precipitating aluminum hydroxide from aluminium chloride by ammonia. This was filtered and roasted in a muffle at about 1200° F. for 2 hours. The tests were carried out exactly as with the ferrates. The results were:—

Table III

Time	Temperature	Zinc Combined
1 hr.	1600° F.	82.2%
3 "	1600° F.	85.3%
6 "	1200° F.	41.0%

These results show that compounds analagous to the ferrates are obtained with alumina, but as this investigation was too wide for the time at our disposal nothing further was attempted, as an experiment made with fireclay and zinc oxide showed practically no combination after 2, 4 and 6 hours at 1500° F; so that the alumina in ordinary ores would not combine with the zinc and lower the extraction. The zinc aluminate was insoluble in dilute sulphuric acid.

Part II

'Sullivan' ore[2] was roasted at temperatures from 1100° to 1500° F. with hand rabbling. At 1100° and 1200°, however, once the sulphur was ignited it was not possible to keep the temperature down until a large amount of the sulphur had been burned off, so that for a time at any rate the whole body of ore was hotter than the registered temperatures, and the particles on the

[2]From the Sullivan Mine at Kimberly, B.C.

surface where the oxidation was rapid were at a high temperature for a short time after each rabbling. This is especially noticeable in the upper floors of the 'Wedge' furnaces after the rabble arm passes; for while the pyrometer in the ore only registers from 1200° to 1300°F. the ore on the surface becomes hot enough to fuse together in small balls. The results are seen in Table IV and Plate 2.

Assay of Ore:—Screen analysis, 70% through 200 mesh I.M.M. Standard.

*	Pb.	Fe.	SiO_2	Al_2O_3	CaO	S	Zn.
	13.2	25.3	2.4	2.9	1.4	28.5	22.6

TABLE IV.

Test	Temp.	Roast			Extraction			
		Time (hrs.)	Total S.	S as SO_4	Zn.	Loss in Weight	Zinc in Tails	Extraction
	F.							
1	1100	7½	6.95	4.6	23.7	39.1	10.2	74.0
2	1200	4	5.45	4.3	23.0	43.4	9.6	76.1
3	1200	4	6.3	5.4	23.0	38.0	9.2	75.1
4	1300	4½	5.35	4.35	23.0	36.0	10.8	70.6
5	1300	4	4.8	4.4	22.8	29.0	10.3	68.0
6	1400	3¾	4.75	4.35	23.6	32.6	12.5	64.4
7	1500	4	5.05	4.31	23.8	27.4	11.5	65.0

TABULATED RESULTS OF ROASTING AND LEACHING TESTS.

In order to test the ferrate theory to see whether a greater percentage of the zinc would be soluble in an ore containing less iron than the ordinary 'Sullivan' zinc ore, three samples, were hand-picked from a carload of the mine shipment before crushing. These samples assayed as follows:—

No.	Pb.	Fe.	Insol.	CaO	S	Zn.
1	5.7	11.3	1.0	0.4	31.7	46.2
2	15.2	10.0	0.4	0.3	30.1	41.6
3	10.4	10.4	0.6	0.5	30.8	44.5

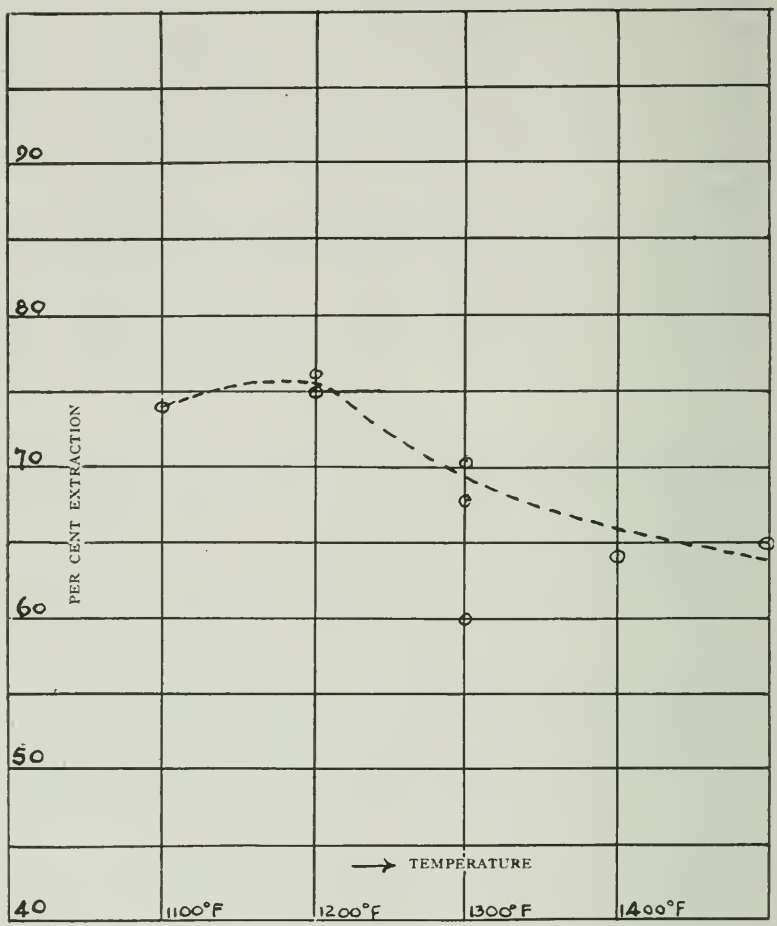


Plate 2.—Curve showing extraction of zinc from Sullivan ore roasted at different temperatures.

These three samples were crushed to 70% through 200 mesh and roasted for 4 hours at a maximum temperature of 1400°+ F. and leached. A sample of 'Sullivan' concentrate from the Ding magnetic machine (—60 mesh + 80), assaying 20% Fe

and 27.2% Zn.was also roasted. A good extraction was obtained —Test 12. The results are seen in Table V.

TABLE V.

Sample	Test	Temp	Time (hrs.)	Roast		Zn.	% Loss in Wgt. in Leaching	Zinc in Tails	Extraction.
				Total S.	S as SO_4				
1	9	1400	4	5.8	5.4	44.1	64.7	19.2	84.5
2	10	1400	4	3.8	3.6	43.9	50.0	15.9	83.2
3	11	1400	4	5.5	5.3	45.9	62.2	14.2	88.4
4	12[1]	1400	4	4.8	4.4	28.2	41.9	8.5	82.5

[1] Ding concentrate.

The size of the roasted particles has little or no influence on the extraction. This can be seen by the tests made on bucked and unbucked samples (Table VI). The unbucked material was leached as it came from the roaster, while the bucked sample was ground as finely as possible in a mortar.

TABLE VI.

Test No.		Loss in Weight %	Zinc in Tails %	Extraction %
9	Bucked	64.8	19.6	84.5
	Unbucked	64.7	19.2	84.5
10	Bucked	49.9	15.5	83.5
	Unbucked	50.0	15.9	83.2
5	Bucked	29.0	10.6	66.0
	Unbucked	28.8	10.5	68.0
12	Bucked	38.7	7.3	84.4
	Unbucked	41.9	8.5	82.5

Roasting and leaching tests were made on some other ores and it may be of interest to examine the results obtained from these ores in connection with the amount of ferrate formed.

The ores treated were as follows:—

	Pb.	Fe.	Zn.
(a)	tr	7.4	47.6
(b)	3.5	15.8	33.0
(c)	1.2	6.0	44.0
(d)	3.0	6.2	44.2
(e)	6.1	4.8	41.0
(f)	tr	9.1	45.1%
(g)	tr	5.3	38.6%
(h)	tr	9.9	39.9%

Leaching tests on these ores gave the following results: temperature 1400° + F. and 4 hrs. roast:

TABLE VII

	Roast		Zn.	On Leaching Loss in Weight %	Zinc in Tails %	Extraction %
	Total S	S as SO_4				
a	2.9	2.6	51.6	70.5	16.4	90.6
b	5.7	5.5	35.7	51.5	15.8	78.5
c	7.4	6.8	44.5	69.8	10.7	92.7
d	8.4	8.2	44.5	71.9	11.2	92.7
e	7.8	7.4	42.6	68.2	11.2	91.9
f	8.6	7.7	46.2	71.7	16.9	89.6
g	8.1	6.9	38.8	55.6	9.1	89.2
h	7.8	7.3	41.5	66.6	12.3	90.1

COMPARISON OF ACTUAL EXTRACTIONS WITH RESULTS TO BE EXPECTED FROM THEORETICAL FORMATION OF FERRATE.

In this comparison those roasts will be considered that were made for 4 hours at a temperature of from 1400° to 1500° F. Each unit of iron will combine with .58 units of zinc according to the formula $ZnO\ Fe_2O_3$, and after 4 hours at 1400° the reaction will be 90 per cent complete, so that we can calculate from the analysis of the ore the amount of zinc that should be insoluble after roasting. Taking 100 grams of ore as a basis and allowin

for the amount of zinc that would probably be present as sulphide and as such insoluble, we get the results tabulated in Table VIII.

TABLE VIII

Sample No.	Fe. in ore	Zn. in ore	Zinc with S grams	Insoluble Zn. grams	Theoretical ferrate grams	Difference, grams	Ratio Zn:Fe.	Actual Extraction	Theoretical extraction corrected for Zn combinedwithS
9	11.3	46.1	0.7	6.06	5.90	+0.16	4.06 :1	84.5	85.0
10	10.0	41.6	0.3	7.10	5.23	+1.87	4.16 :1	83.2	87.4
11	10.4	44.5	0.3	5.06	5.30	—0.24	4.28 :1	88.4	87.8
12	20.0	27.2	0.6	4.35	10.40	—6.05	1.36 :1	82.5	61.0
(6 & 7)	25.3	23.0	0.5	8.05	13.20	—5.15	0.92 :1	65.0	42.2
a	7.4	47.6	0.5	4.35	3.87	+0.45	6.43 :1	90.6	91.5
b	15.8	33.0	0.3	7.35	8.25	—0.90	2.09 :1	78.5	76.1
c	6.0	44.0	1.0	3.13	3.13	0	7.3 :1	92.2	90.7
d	6.2	44.2	0.3	2.85	3.24	—0.39	7.13 :1	92.7	92.1
e	4.8	41.0	0.6	2.96	2.60	+0.36	8.55 :1	92.0	92.0
f	9.1	45.1	1.8	3.98	4.75	—0.77	4.74 :1	89.6	85.8
g	5.3	38.6	2.2	1.84	2.78	—0.84	7.26 :1	89.2	87.2
h	9.9	39.9	0.8	3.25	5.17	—1.92	4.02 :1	90.1	85.6

From an examination of these figures, it will be observed that the amount of insoluble zinc present after roasting is very close to the theoretical amount that should be formed as zinc ferrate with the exception of the regular 'Sullivan' zinc ore (Nos. 6 and 7) and the magnetic concentrates (No. 12). In each of these instances the ratio of zinc to iron is close to 1:1; whereas, in the others, with the exception of one sample, it is at least 4 to 1. In some way, therefore, in these samples, the combination is prevented from being as complete as one should expect from the composition of the ore. One explanation may be that at this concentration of the iron and zinc another compound may be formed containing a greater ratio of iron to zinc than that represented by the formula $ZnO\ Fe_2O_3$, but we have been unable to prove this and regard the existence of any combination except $ZnO\ Fe_2O_3$ as unlikely.

On the other hand it may be that free crystals of pyrrhotite do not combine as readily as those crystals which may be attached directly to crystals of blende or as that iron which may be isomorphous with the zinc in the blende. One test made with the

addition of 20% SiO_2 to the 'Sullivan' zinc ore before roasting indicated that the addition of some inert matter decreases the amount of insoluble zinc formed, as an extraction of 76% of the zinc was obtained in this case as against 65% without the addition of the silica.

The results of the above test were as follows[3]:—

Roast		Zn.	Loss in Weight	Zinc in Tails	Extraction
Total S	S as SO_4				
3.6	3.2	19.0	34%	6.7%	76.7

[3]Compare tests 6 and 7.

One could, of course, determine the effect of adding pyrrhotite or pyrite crystals to the ore by observing whether the extraction were decreased or not, but this has not been attempted as the results embodied in the present paper, as far as they go, explain the reason for the failure to obtain a high extraction on the ore treated at Trail.

Summary

1. It is proven that zinc oxide and ferric oxide will combine readily at temperature above 1200° F., the amount of combination depending principally on the temperature and the time.

In the roasting of the ore in the Wedge furnaces, however, in order to desulphurize the ore sufficiently, it is not possible to keep the temperature down to a point at which the formation of ferrate does not occur. In the upper part of the furnace especially, it is difficult to keep the ore from becoming too hot owing to the large amount of sulphur present, but it is possible that by decreasing the frequency of the rabbling the ore might be kept for a longer period in the furnace and subjected to a lower temperature, as the more rapid the stirring while the sulphur is burning readily the higher the temperature attained. In connection with this, in the case

of the 'Sullivan' ore as at present roasted, the maximum amount of ferrate seems to be formed, as some of the regular roast was heated to a temperature of 1500° for 5 hrs. and the extraction on this material was no worse than on the original.

2. Free alumina combines with zinc oxide, but when present as a silicate the amount of combination is negligible.

3. There are four ways in which a better extraction of zinc might be obtained from an ore high in iron.

a. By roasting at a low temperature. (The remarks made at the beginning of the summary apply to this point.)

b. By eliminating the iron before roasting.

c. By the addition of some substance to the ore which would prevent the combination of the two oxides.

d. By finding some solvent for the zinc would leave the iron oxide undissolved.

THE KINGDON LEAD MINE

By John E. Hardman

Annual Meeting, March 1917

For many years there has prevailed in this Dominion an opinion that Eastern Canada was devoid of mines of base metals beyond the phenomenal nickel deposits of Sudbury, the iron ores of Nova Scotia and Newfoundland and the sulphur and copper deposits of the Eastern Townships. Credit is given for a small production of gold in Nova Scotia, but if one wanted really large and permanent deposits of copper, lead or zinc he was supposed to waste no time in considering *Eastern* Canada, but to immediately start for our Western-most Province, where such things were to be had "in every mining camp."

The phenomenal successes of the two Ontario districts, (Cobalt and Porcupine) in silver and gold during the last decade drew much attention to that Province, and its resources in base metals were, and are yet, the subject of economic study by men who have faith that the geologically older formations in Eastern Canada contain stores of metallic wealth that, when properly exploited, will be able to compete successfully with Western production.

The Kingdon property is one of the comparatively small, galena-bearing properties that are characteristic of the rocks called the "Grenville Series" by Sir Wm. Logan in 1845. Its discovery was due to a farmer in the years when the lead-bearing veins of Frontenac, Leeds, Bannockburn and other districts were arousing high hopes of a profitable lead industry, later doomed to disappointment. Some time in the seventies it was purchased by the late James Robertson of Montreal and has remained in his estate ever since.

In the early part of 1914, the writer was called in to advise concerning the Kingdon property, to assist in determining whether it should be written amongst 'Assets' or whether it should be charged finally and forever to 'Profit and Loss.' Previous examinations by other engineers, some of whom were members of this Institute, were either adverse or non-committal.

The first examination of this property showed its development to be quite inadequate to form the basis of a permanent

opinion, there was but one shaft which had a depth of 50 feet and there were only two other shallow pits.

The published, or accessible, views of geologists upon these deposits of the Grenville Series were not favourable to continuance of the ore bodies, the general view taken being that they were segregations of limited extent. It was, therefore, recommended that a certain amount of work, laid out to prove continuity, should be done before a decided opinion was expressed.

This was immediately granted, and during the summer of 1914 the preliminary work was finished and an opinion given that fully 20,000 tons of ore were available of a general average content of 10%; this gave, at once, a valuation to the property that was satisfactory.

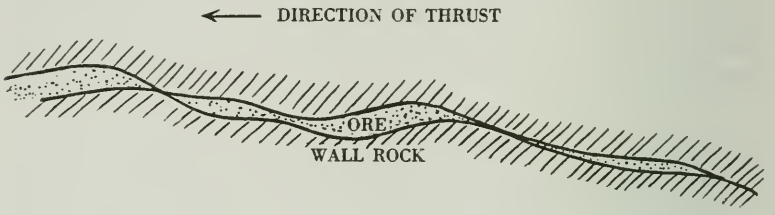
From this beginning (in the month of September 1914) to the present time the deposit has continued to increase in value, and since the commencement of smelting operations in the first week of October 1916 the property has returned all monies expended, and has no charge on the debit side of the ledger.

The property is located on an Island called 'Laflamme' or 'Chat Island' in the township of Fitzroy, and Province of Ontario. The land is made an island by a small branch of the Mississippi River which forks about one mile below Galetta, one channel sweeping to the North the other to the south, and both flowing into the Ottawa River. The elevation of this island nowhere exceeds 50 feet above the level of the river.

The rocks of the island, and of the surrounding shores are that series of schists, micaceous, hornblendic and calciferous, interbanded with ancient limestones, to which Sir Wm. Logan gave the name of the 'Grenville Series.' These beds of highly crystalline limestones and of schists are conformable, though frequently crumpled, and are tilted to a vertical or nearly vertical (74° to 82°) position. Cutting these beds, usually at right angles, but sometimes almost in the line of strike are a few dikes of eruptive rocks which have been described by learned lithologists as diorites.

Previous examiners described the strike of the ore body as uniformly N.W. and S. E., and as conforming with the strike of the "containing crystalline limestone," but development has

shown that the ore body follows a line of fracture which is distinctly serpentinous when plotted, and that the larger ore bodies fill the spaces which are due to a second thrust that occurred *after* the shearing, and whose direction was that of an end thrust. The following figure is from the survey of the 100-ft. level. It is believed that failure to recognize the waving line of the fracture and consequent varying direction of the deposit is the reason for some previous examiners reporting that there were "two or more" veins intersecting each other.



In the heading of the first level excellent photographs have been obtained, showing clearly the line of fracture, or mark of "discission," thus effectively proving the existence of a main fracture.

The ore deposition is of crystalline galena occurring, in the wider portions, in masses from 12 inches to 30 inches broad, but also as ore disseminated along the walls; I use the plural "walls," but it is not often that both walls can be clearly seen, only the foot-wall is always in evidence. On the walls occasionally is found from half an inch to three inches of 'gouge' which analysis shows to be 50% $CaCo_3$. Graphite, in minute specks and in very small quantity, is noticed on or near the wall of the fracture. Slickensides, or striations on the wall, are much in evidence.

Associated with the galena are small amounts of the usual minerals, pyrite and blende, and very rarely some chalcopyrite. The galena is not argentiferous, the average of many assays giving but one-third of an ounce to the ton of ore. The average silver value of the *bullion* produced is about 4 ounces of silver to one ton of lead.

So far it has been noted that where the vein is richest in galena it is poorest in iron and zinc, an occurrence which has been very beneficial to the furnace department.

The general average contents in metallic lead of the whole length exploited (550 ft.) has been about 8.5%, as determined by weight of the pig lead produced. The proximity of the eruptive dikes does not appear to materially influence the metallic contents of the vein.

Some 1400 feet to the northwest of the portion of the vein which has been worked is an outcrop from 3 to 4 ft. in width which has been prospected to a depth of about 11 feet; there can be little doubt that this is the extension north-westwardly of the vein developed at the mill.

The vein is opened and worked by overhand stope methods; the ventilation is good, but the amount of water encountered is heavy and is a serious item in shaft sinking. This however will, it is hoped, be overcome by the new development to be undertaken this summer, which will consist of a vertical shaft in the country rock, opened into the vein by cross-cuts below the vein workings.

The property includes some 1900 acres, upon which the deposit has a possible length of 3500 feet; of this length about 550 feet has been opened by underground levels at 100 feet and 185 feet in depth; surface openings cover 1800 feet of the possible 3500.

Mill.—In the late summer of 1915 a concentrating mill was erected on the property with a capacity of 75 tons of ore in 24 hours.

The problem of concentration at the Kingdon is an easy one, the only mineral desired to be saved (galena) has the specific gravity of 7.4, the gangue is 2.6 and the specific gravity of the polluting minerals varies from 4 to 5. On account of the friability of the galena and its frequent occurrence in quite large masses, the mill was designed to separate the galena in as large particles as possible and as soon as liberated from the matrix of calcite.

The polluting minerals (blende and pyrite), being also very friable, their separation from the galena was left entirely to the James tables, which perform the work very satisfactorily.

The ore is first crushed by rock breaker to about $\frac{3}{4}$-inch mesh, then passed through a screen with meshes 6 mm. in diameter. The oversize is returned to crushing rolls and reduced to 2 mm.

The undersize is fed to a Newaygo screen where all material over 2 mm. is separated, the oversize going to two Richards pulsating jigs, the undersize joining the product from the rolls, and the combined fine material going to an 8-ft. Callow tank. The Richards pulsating jigs produce a most excellent concentrate, the assay values of which vary from 82% to 85.9% of *metallic lead.*

The thickened pulp from the Callow tank is passed over a Richards pulsating classifier, the sands going to James tables and the slime to a James slimer.

The *highest* assay yet obtained from the tailing is less than ½% of lead; the *average* percentage of loss in the mill tailing is 3/10ths (0.3) per cent of lead.

The grade of concentrates made is unusually high; the average percentage of lead in the table products is 80+, in the jig headings 82%. From the mill the concentrate is sent by tramway to the smelter.

The process used at the Kingdon mine for the production of metallic lead from high grade concentrates is novel in this Dominion. It was the writer's privilege to attempt to solve this problem, and the solution was found in the 'Newnan' improved hearth furnace and its accessories, as developed and put into commercial use by Mr. W. E. Newnan, the able Superintendent of the St. Louis Smelting and Refining Company at East St. Louis.

The old 'Scotch hearth' and its Americanized 'Moffet hearth' were limited in capacity, injurious to health and fatiguing to the hearth men. The rabbling by hand required men of strong and heavy physique, the loss in fumes was high and the production of pig lead from 50% to 55%. Mr. Newnan's inventions and improvements have obviated all these troubles; the mechanical rabbler with the modified form of furnace have removed all severe muscular labour; the improvements have cut down losses in metal, in fume and in slag, and it is now possible to recover fully 91% of the metal in the ore treated. Mr. Newnan has published his results in a recent paper contributed to the Transactions of the A. I. M. E. (October, 1915).

The entire rights of this furnace and process in Canada have been acquired by the Kingdon Mining, Smelting and Mfg. Co.

View of furnace house, goosenecks and flue

Since the beginning of smelting operations in the first week of October 1916, there has been no case of illness due to lead fumes or lead poisoning. Careful accounts kept have shown a recovery *in the form of pig lead* of 91% of all lead fed to the furnace; the recovery of lead fume and dust from the goose-neck chambers and the bag house makes the total recovery up to 96%.

The pig lead produced is of unusual purity; as the lead issues from the furnace it falls directly into moulds holding 100 lb. As these pigs cool they are removed to the remelting kettle

A nearer view of furnace house, showing goosenecks

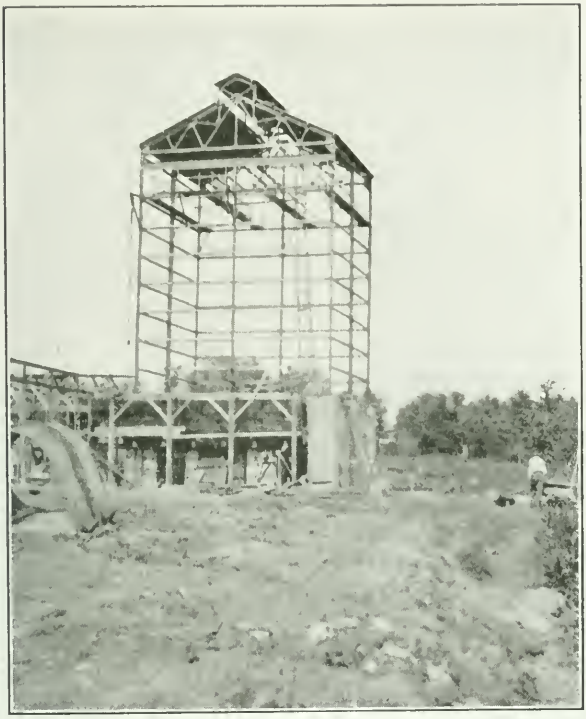
Bag-house and flue

which holds 10 tons to a charge. Here the first metal is re-melted and skimmed; it is then treated to remove zinc, copper and antimony, and is poured into 60 lb. moulds and shipped, when cooled, to the market.

The lead thus produced will roll, without further purification, into sheet lead, and has been used for acid chamber lining without disappointment in any particular.

The great World War, which we all hope is shortly to be concluded with victory for human rights and freedom, has been

A view of the flue



Skeleton of bag-house

full of bitter experiences to most of us in Canada, yet it has been, I venture to say, of inestimable value to Canada as a nation, for undoubtedly it was the cause of Canada's renaissance as an industrial community. For years we have been exporters of crude materials; for years we have heard of the futility of attempting to refine our metal products, and, but for the War, it is altogether probable that we would still be shipping our copper as matte or anodes, our zinc as ore, and our lead as crude pig, or unrefined, metal. Now we may safely say that our day as exporters of crude material has vanished, that we are now competent to refine, and *are* refining, so that our production is to be reckoned as the finished product. And among the small but important factors in this renaissance has been the demonstration that *some* of the neglected mineral deposits of Eastern Canada can be made profitable under wise guidance. One proof is the record of the Kingdon lead mine.

FURTHER NOTES ON YUKON MINING PROBLEMS

By Henry M. Payne, New York.

Annual Meeting, Montreal, March, 1917

At the annual meeting of the Institute in 1913, the writer described conditions to be met in the Yukon, outlined the problems which must be solved in order most effectually to recover the values lying in the frozen gravels, and detailed the research work which up to that time had been conducted with a view to placing thawing on a scientific basis.

The primary results of that work were a classification of the materials to be thawed, and a knowledge of their temperature, specific heat, specific gravity, and voids frozen and thawed. Secondarily, experiments were made to determine the rate of natural or climatic thawing. The results of this research work, as outlined four years ago, have amply justified the undertaking, and in general, the predictions then made, have been fulfilled and exceeded.

Fig. 1
Old Steam Point and New Drill Bit Point

It is now a recognized fact that wherever the surface can thoroughly be cleared, ploughed, ditched and drained, natural thawing rapidly ensues, and if the overlying muck be ground sluiced, a period of about three years will thaw to bedrock at medium depth in the main valleys. This ground sluicing not only facilitates thawing, but also effects a stupendous reduction in non-auriferous yardage which must otherwise pass through the dredge, with its consequent wear and resulting replacements, and also reduces the average annual consumption of power by at least 1,500,000 k.w.

Fig. 2
Pot Holes After Thawing with Steam.

The introduction of hot water thawing on a commercial scale, was made in the summer of 1915, with points spaced 16 feet apart in triangular arrangement, the boilers being fed by injectors and the main line taking its supply from the blow-off pipe. The great loss formerly resulting from condensation of steam was immediately corrected and the thawing period materially decreased. The next obstacle to be overcome was the difficulty of driving the thawing points to bedrock at one driving, in order that the thawing process might logically proceed upward from the bottom, rather than at irregular intervals from the

surface down, as was necessarily the case with the regular steam points in use.

To accomplish this, an Ingersoll-Rand drill bit point was welded onto the end of the three-quarter-inch-steam point and a three-thirty-second-inch-hole was drilled at the top of each of the four flutes. Thus, instead of one three-sixteenth-inch-hole at the end, as in the old point, we have five holes, four in the sides and one in the end, and as a result, it is possible to drive the point directly through any boulders which may be encountered, as the frozen boulder, when partially drilled

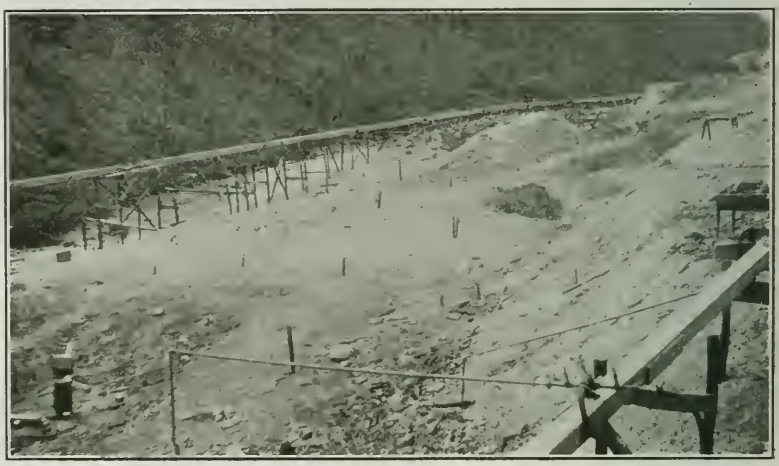


Fig. 3
Complete Subsidence of the Ground After Thawing with Hot Water.

through, expands from contact with the hot water and splits, allowing the point to drop on down to the next boulder.

The hot water in the meantime has a sluicing effect from the side holes which could not be obtained with the old point, having one orifice only, and the thawing process and removal of fines around the point proceeds very rapidly. Also, it will be recalled, an average of 46% of the frozen material *in situ* is ice, which as it melts, loosens the gravel and boulders, so that when the thawing process has started at bedrock, a rapidly increasing

subterranean cavern is formed, and as the level of the underground lake of hot water rises, the overhanging gravels are loosened and spall off.

Thus in a comparatively short time, the entire valley, from rim to rim, has been thoroughly thawed, and has subsided uniformly, instead of leaving pot holes around each thawing point, as was the case when steam was used. It was found by experiment that the thawing period was two and a half times as long under the old method, as under that described.

A further incentive to the use of hot water, is the fact that

Fig. 4
Anvil for Driving the Drill Bit Points

with steam, whenever the pressure is allowed to lessen at the boiler, the vacuum created in the network of the steam lines, immediately causes many of the points to fill with soft mud, and choke, so that even when pressure has been resumed, the points will not blow free, but must be pulled and cleaned and re-driven. With hot water as a thawing medium, the worst that can happen is that cooler or cold water is passing into the ground, and an increased consumption of fuel and a lengthened thawing period will result.

Standard time periods were determined for thawing any

given material at any depth, so that the whole thawing process is now reduced to a scientific basis.

For hot water thawing, with a point of the type described, the point should remain at the maximum depth four-sevenths of the total predetermined period. It should then be drawn and another point two-thirds as long inserted, and left for two-sevenths of the thawing period. If there is no muck, and the frost line is 8 feet or more below the surface, this will probably suffice; otherwise a point one-third of the length of the original one should be left for the remaining one-seventh of the required total thawing period.

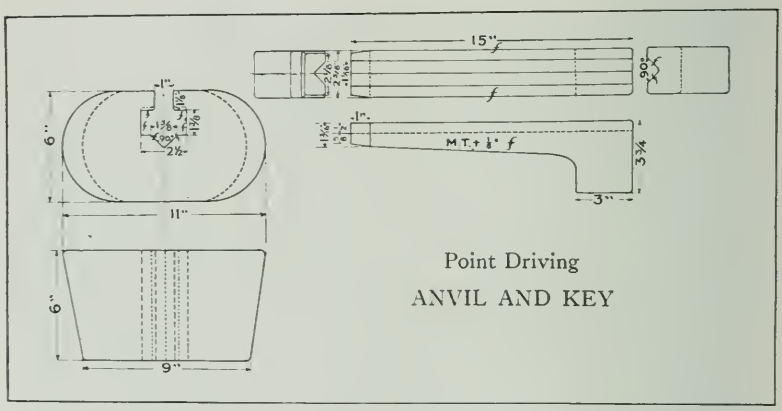


Fig. 5
Designs of Anvil and Key

To eliminate the use of ladders and the breaking of points, and to accelerate the rate of driving, anvils weighing about 100 lb. were forged from old dredge bucket pins, slotted to pass over the thawing point, and keyed in place. Handles are inserted in each side, and the helper turns the point, as in ordinary rock drilling, while the operator stands on the ground and drives the point until the anvil reaches the ground, when the key is knocked out, the helper raises the anvil to a convenient height, the key is again inserted, and the operation repeated, until the desired depth has been reached.

During the entire season's operation, not a single point was broken in driving by this method, and where no boulders

were encountered, the weight of the anvil alone, was sufficient to force the point to bedrock, when twisted back and forth.

Since the points are placed in regular arrangement at fixed distances from the supply pipe line, rubber hose are used only during driving, and standard pipe connections are put on when the driving operation is completed. Between these pipe connections and the main line, ordinary railroad-train hose couplings are inserted, obviating the earlier difficulties with leaky unions, reducing the time necessary to make or break a connection, and eliminating the Stilson wrench as a necessary factor.

The following comparison is given as the result of a season's work on adjoining sections of the same creek under exactly analagous conditions, with both steam and hot water.

	Steam Plant	Hot Water Plant
Driving time of points	9.8 ft. per hr.	18.3 ft. per hr.
Area thawed per point	64. sq. ft.	231.7 sq. ft.
Quantity thawed per point	1,419.7 cu. ft.	5,253.8 cu. ft.
Area thawed per point-hour	.76 sq. ft.	1.62 sq. ft.
Quantity thawed per point-hour	16.87 cu. ft.	36.8 cu. ft.
B. T. U. per point-hour	63,800	117,000
B. T. U. per sq. ft.	84,000	74,389
B. T. U. per cu. ft.	3,780	3,274
Point-hours to thaw 1,000 cu. ft.	59.2	27.2
Fuel Consumption		
To Thaw 100 sq. ft.	1.29 cords	.44 cords
To Thaw 2,000 cu. ft.	1.17 cords	.39 cords
Points carried per H.P.	.516	.533
Sq. ft. per H.P. hr.	.392	.69
Cu. ft. per H.P. hr.	8.7	15.7
Number of points per unit area	100%	30%
Piping, trestling, etc	100%	52%
Ladders	100%	1 only
Rubber Hose	1 per Point	Driving only.

It will be observed that the B.T.U. per square foot and per cubic foot were less with hot water than with steam, the reason for this being that with steam thawing, under the old method, much heat was dissipated in the thawed ground at the top.

In building the pipe line for hot water, stakes driven low in the ground are all that are necessary, as 'dry steam' is no longer an objective.

The labour saving on a single unit thawing plant, would be as follows:

Standard steam thawing crew of sixteen point men on day shift and eight point men on night shift. Total 240 man-hours. Average *net* square feet thawed per day = 2,348.8.

To thaw the same area with hot water would require 2,348.8/231.7 = 10 points, *net*.

Fig. 6
Steam Thawing Plant

Two pairs of point men drive 5 drill bit points each with an anvil, in ten hours, viz:

Driving	6 hours
Pulling	1½ "
Connecting up	1½ "
Miscellaneous	1 hour
	10 hours

Two additional men to bar for frost, and two men to look after pipes and fittings, and straightening points. Total 80 man-hours.

240 − 80 = 160, man-hours @ 60c. an hour for wages and board = $96 per day per crew, saved.

In other words, eight men working day shift only, with four boilers on hot water, thaw the same area per day formerly thawed by sixteen day men and eight night men, with steam.

The fuel saving is at least 5 cords per 24 hours. It has also been found that the ground, after thawing with hot water, is

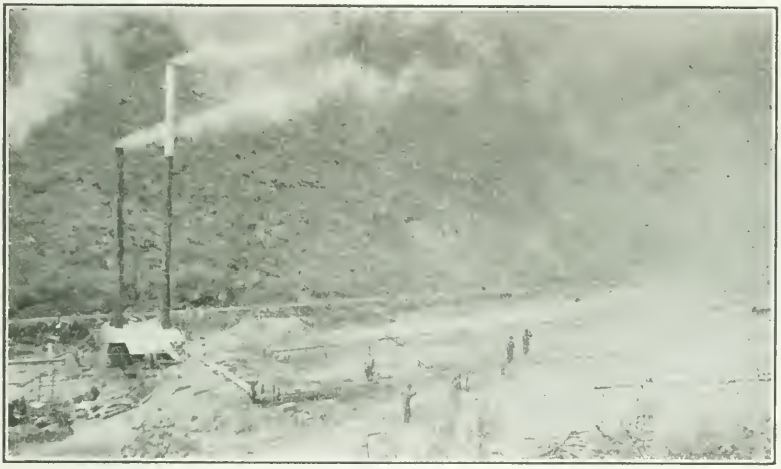


Fig. 7
Hot Water Thawing Plant

more uniformly heated than with steam, and less trouble with 'back freezing' occurs, so that one dredge operating during the season of 1915 did not close down until late in January 1916, and could have continued in operation all winter, notwithstanding the low temperatures, but it was thought best to shut down for general overhauling for the succeeding season.[1]

The Klondyke type of dredge, as differentiated from the California type, is a development distinctly indicated for use in

[1] Since this paper was prepared, the writer is advised by the Canadian Klondyke Mining Company, as of February 9th, that Number 2 Dredge was still in operation this year with an excellent chance of establishing a twelve month dredging record, notwithstanding the fact that December 1916 and January 1917 were the coldest two months since 1902.

operating Arctic gravels. Its efficiency is high, and its operating cost and upkeep relatively low, in proportion to yardage handled.

The average consumption of power varies from .75 to 1.5 k.w. per cubic yard dug, depending on depth and amount of boulders and old timbers encountered. In 1915 a record of 23 hours 37 minutes actual operating time, out of a possible 24 hours, was established. The use of cocoanut fibre mats with expanded metal riffles on top, rather than of quicksilver, has been justified by

Fig. 8
Emergency Repair Barge on Power Ditch

the results obtained from extended tests of tailings and sluice washings. The tests indicate a thorough recovery of the gold. The higher price obtained per ounce for gold so recovered is also an inducement not to use quicksilver. It is the writer's opinion, however, that a prerequisite to this method is the large gold saving area afforded by one of these dredges, and that on a smaller type, the usual amalgamating process is necessary.

The power plant is in continuous operation, as described previously by the writer The ditch between this plant and the intake is six miles long, of which $4\frac{1}{2}$ miles are in sphagnous bog and glacial muck, which as it thaws and settles necessitates re-inforcement of the lower bank with brush mats, and filling up to grade. Whenever any weak places occur, double sections

of lapped 2-inch sheet piling are driven, and the ditch and hillside below it are regularly patrolled. Where indications of serious weakness occur, cribbing is built on the lower side and filled by hydraulicking gravel from the hillside above. To accomplish this at a minimum cost, a barge has been equipped with centrifugal pumps and over the top a chute is built, carrying the sluicings across the ditch to the cribbing on the lower side. This barge can be moved quickly from place to place along the ditch and constitutes an emergency equipment instantly available.

With the development of the famous Lena River goldfields in Siberia, an interesting opportunity for comparative studies in climate and precipitation is presented. The region around Bodaibo, Siberia, is very similar to that around Dawson, in the Yukon. Both cities are the centre of a mining district operated or operatable by the same methods, and having the same general problems to be met. Lying in approximately the same latitude, each at the confluence of two large rivers, whose general direction, branches, and sources, are in the same relative position, it might be expected that little if any variation in climate would occur.

To make such a comparison, the Government records of precipitation and temperature have been secured for a series of years and platted. The readings at Dawson were taken twice daily for a 15-year period, and both maxima and minima are given.

The Siberian readings were also taken twice daily, but the published records give only the mean temperature for each day. The data so compiled for the two regions, represent over 47,000 separate items. In the accompanying charts, the yearly variations by months in both temperature and precipitation are indicated, and a composite curve for the entire period is then superimposed on the corresponding curve for the other country. Monthly variations by years are also shown, so that seasonal irregularities may be readily detected.

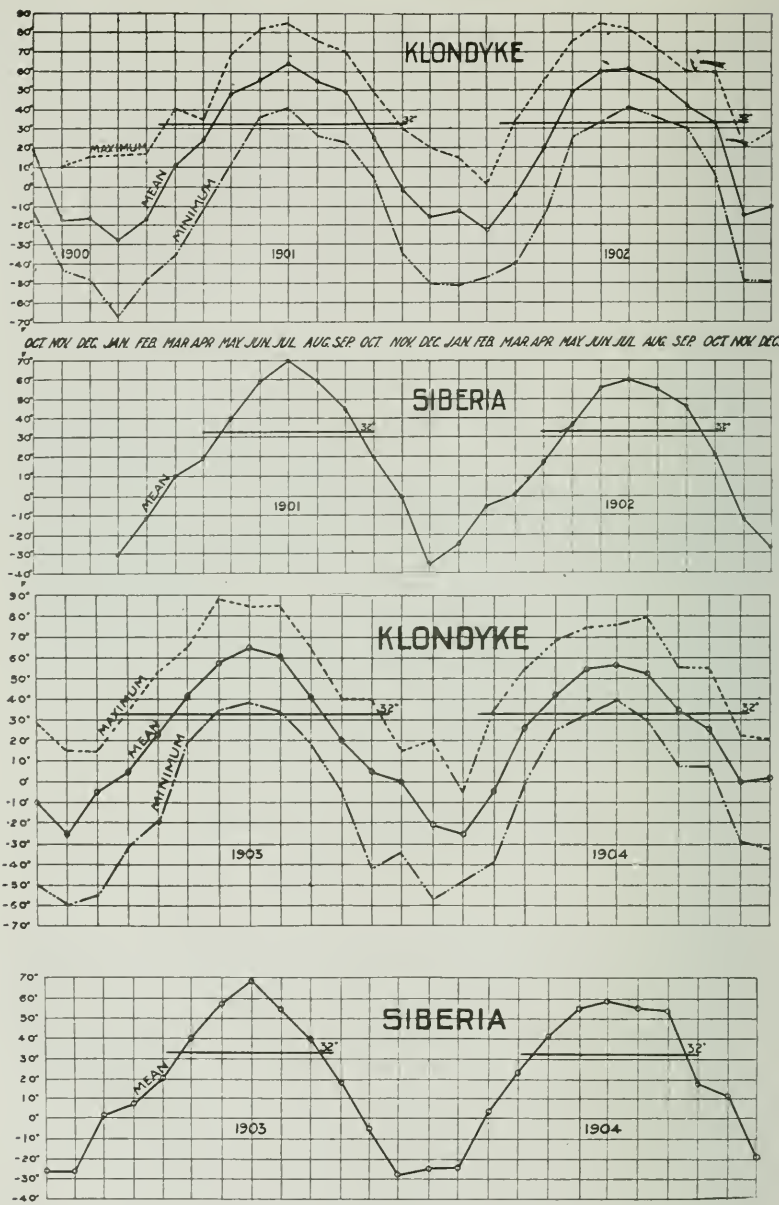


Fig. 9

Temperature variations in the Klondyke and Siberia—1900-1915

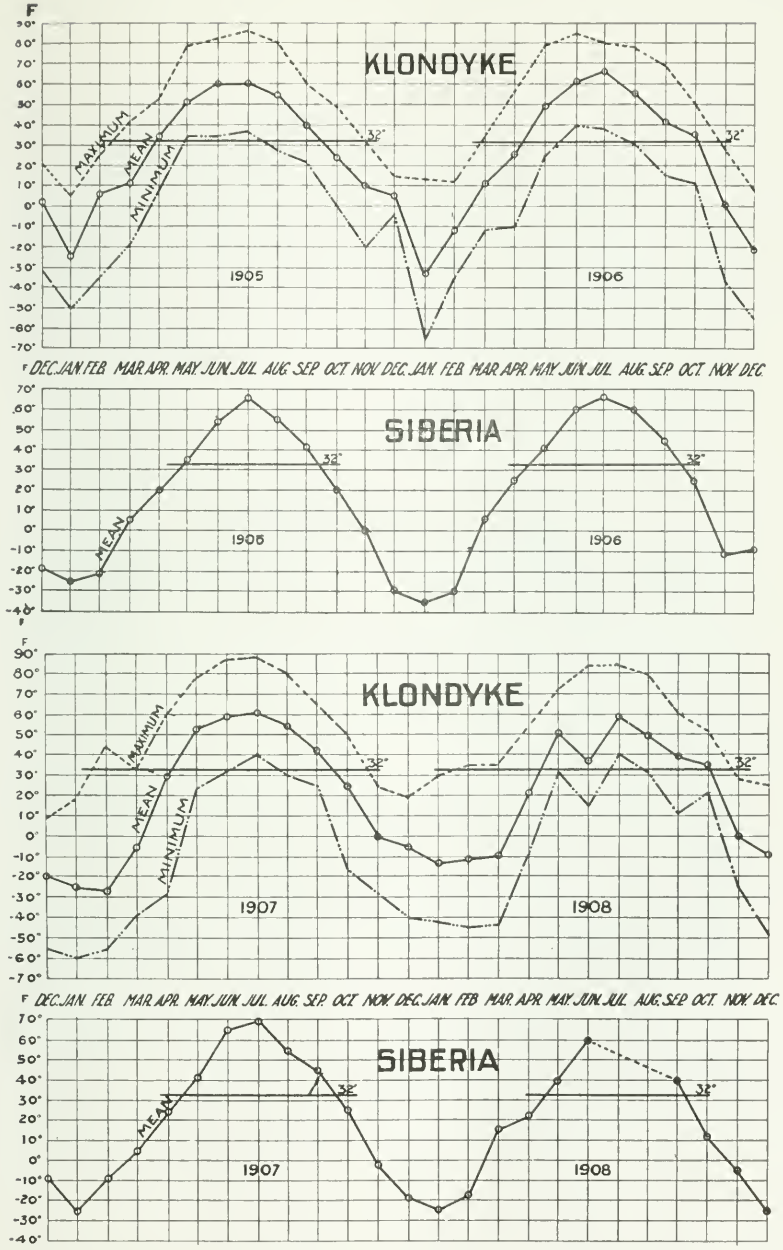

KLONDYKE
MAXIMUM
MEAN
MINIMUM
32°
1905
1906
F DEC. JAN. FEB. MAR. APR. MAY. JUN. JUL. AUG. SEP. OCT. NOV. DEC. JAN. FEB. MAR. APR. MAY. JUN. JUL. AUG. SEP. OCT. NOV. DEC.
SIBERIA
MEAN
1905
1906
KLONDYKE
MAXIMUM
MEAN
MINIMUM
1907
1908
SIBERIA
MEAN
1907
1908

Fig. 9—Cont.

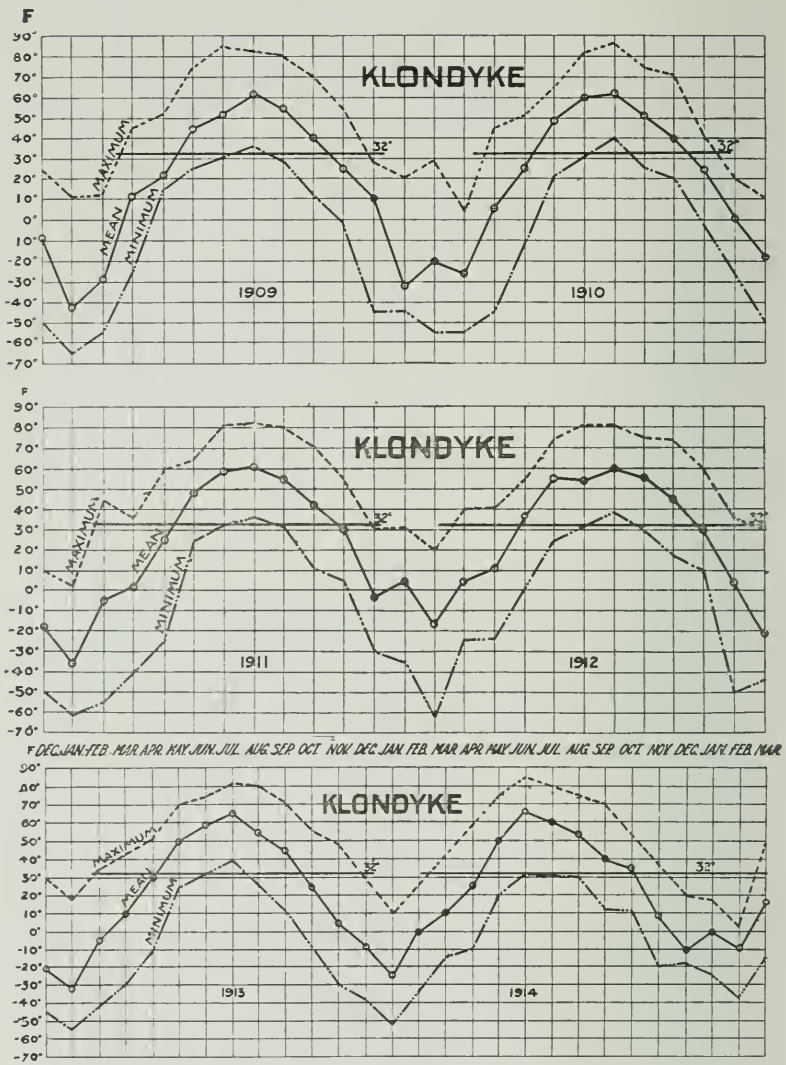

F
KLONDYKE
MAXIMUM
MEAN
MINIMUM
32°
1909
1910
1911
1912
1913
1914
F. DEC. JAN. FEB. MAR. APR. MAY. JUN. JUL. AUG. SEP. OCT. NOV. DEC. JAN. FEB. MAR. APR. MAY. JUN. JUL. AUG. SEP. OCT. NOV. DEC. JAN. FEB. MAR.

Fig. 9—Cont.

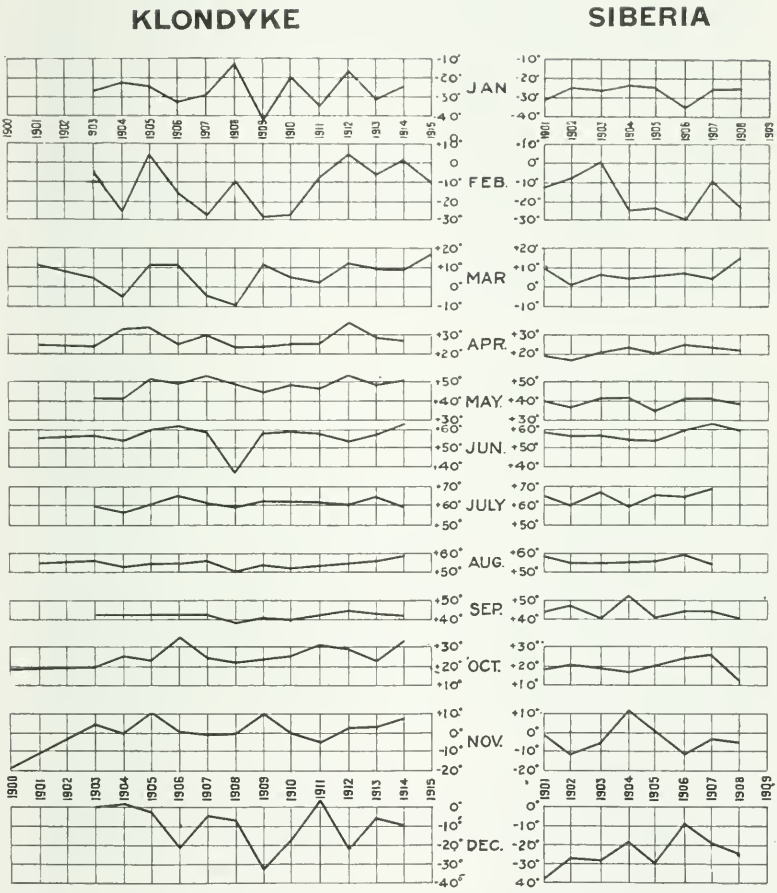


Fig. 9—Cont.
Yearly variation in monthly temperatures

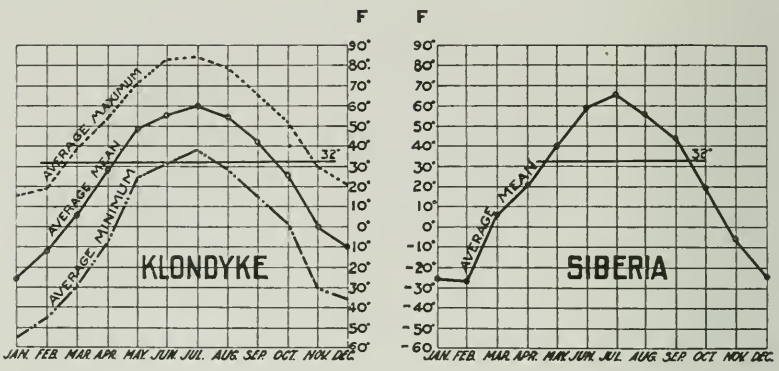
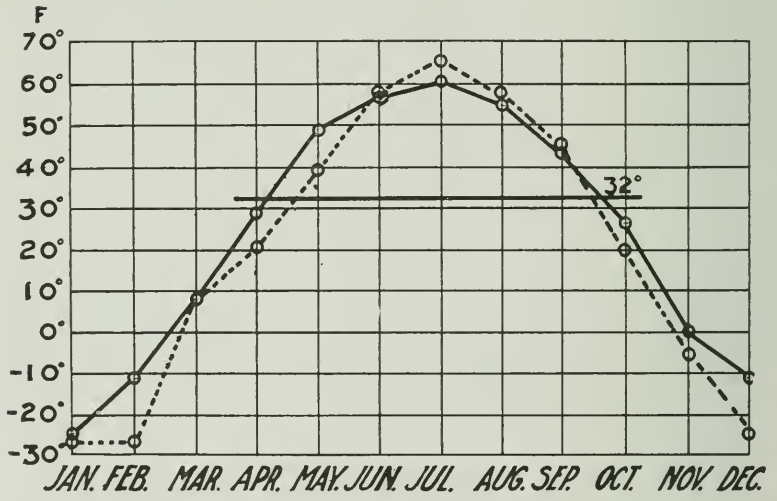


Fig. 9—Cont.



. 9—Cont.

Composite curve showing average mean temperatures for Klondyke ———; and Siberia - - - -

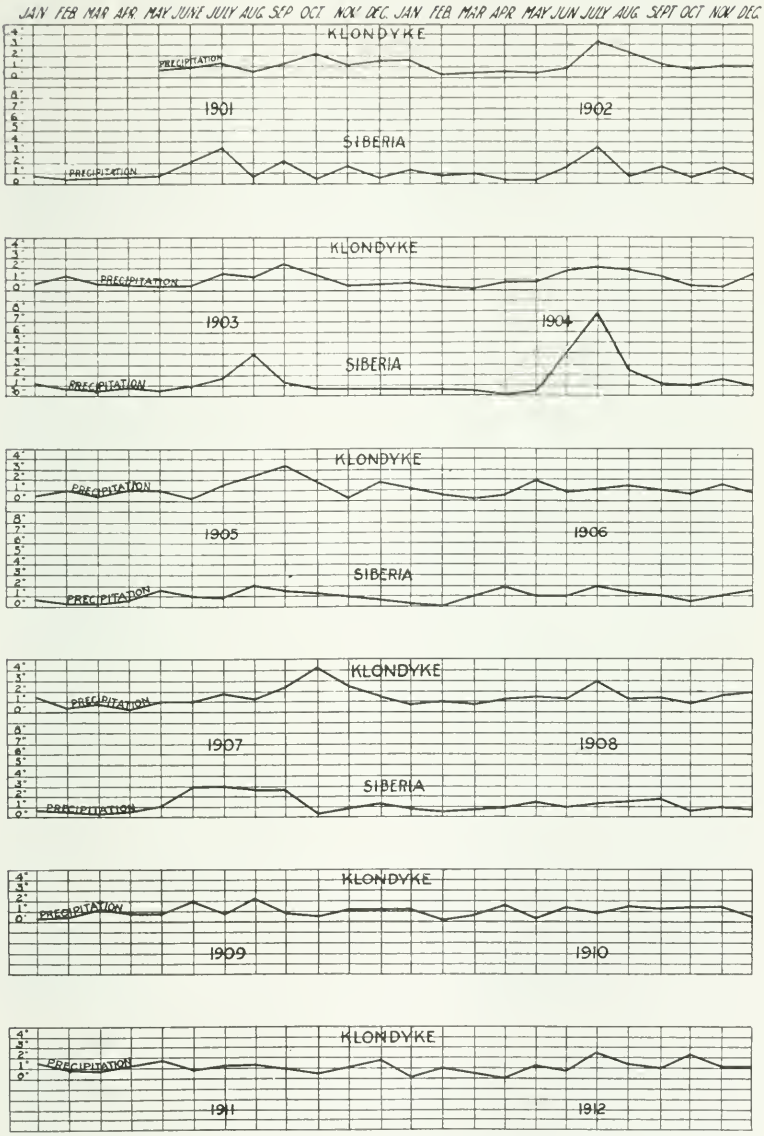


Fig. 9—Cont.

Average monthly precipitations in the Klondyke and Siberia, 1900-1915

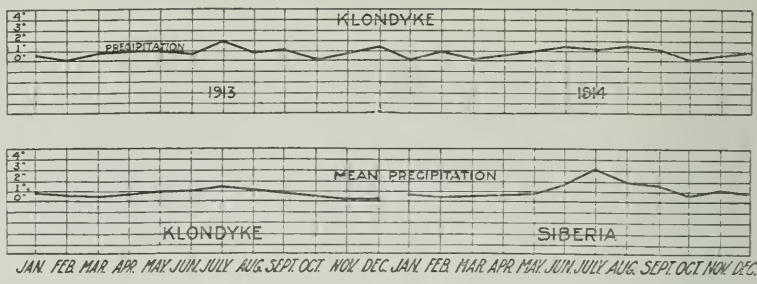

KLONDYKE
PRECIPITATION
1913
1914
MEAN PRECIPITATION
KLONDYKE
SIBERIA
JAN. FEB. MAR. APR. MAY. JUN. JULY. AUG. SEPT. OCT. NOV. DEC. JAN. FEB. MAR. APR. MAY. JUN. JULY. AUG. SEPT. OCT. NOV. DEC.

Fig. 9—Cont.
Average monthly precipitations in the Klondyke and Siberia 1900-1915

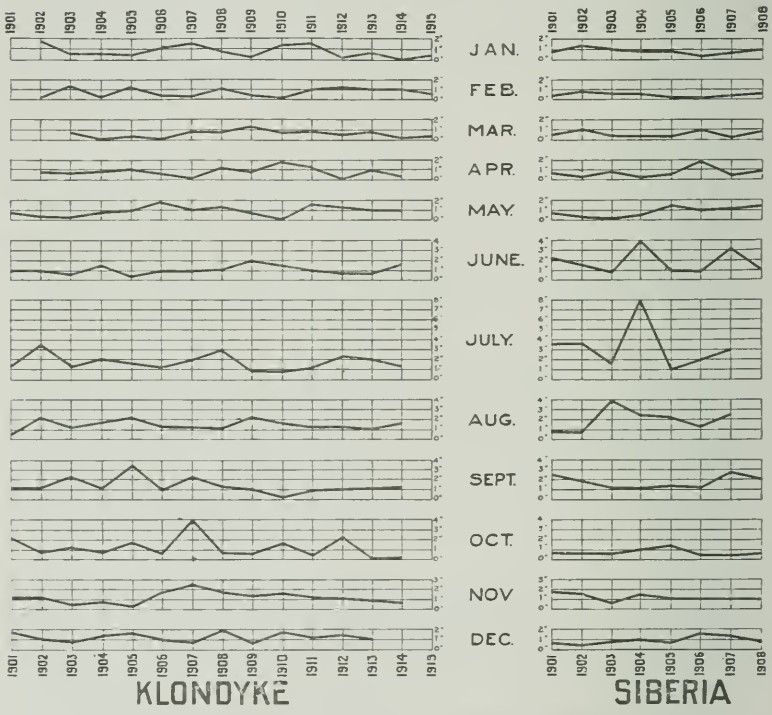
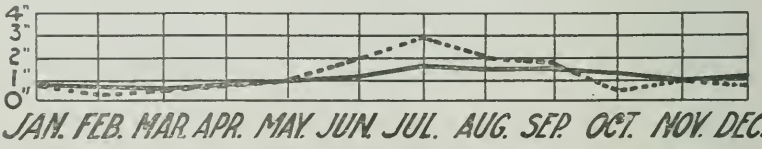


Fig. 9 Cont.—Yearly variations in monthly precipitations



Composite curve showing mean precipitation in Klondyke —— Siberia - - - -

As a result of this comparison the following deductions may be made:—

1.—The Siberian winter is longer than that of the Klondyke, the months of December, January and February averaging almost constantly from −24° to −28° F. while at Dawson the minimum is reached in January and the months of December and February, in the 15-year period recorded, averaged 13° to 17° warmer than the corresponding temperatures for Bodaibo.

2.—March and September in both countries are about the same, but the Siberian summer season while the warmer of the two, is also about one month shorter.

3.—The months during the entire period of years observed, which showed uniform average temperatures, with very few fluctuations, were:—

Klondyke		*Siberia*	
May	48°	April	21°
July	61°	May	39°
August	54°	June	57°
September	42°	August	54°

from which we may infer that the months not named vary greatly from year to year.

4.—No material change in the seasons is appreciable, but there appears to be a tendency in the Klondyke, since 1911, for the months of March, April, May and June to be slightly warmer than in the ten years preceding.

5.—In the matter of precipitation, very little fluctuation occurs in the Yukon, the maximum being in July and the minimum in March.

The Siberian minimum occurs in February and a flood period is reached in July. The total precipitation at Bodaibo, however, is greatly in excess of that at Dawson.

6.—The months in which the greatest irregularity in precipitation occurs in Klondyke are July, September and October; and in Siberia, June, July and August.

7.—Beginning with 1905 the period of maximum precipitation in Siberia appears to have been less marked, and to have extended over three months, each succeeding year showing a wider distribution of rainy season, with less precipitation per 24-hour period, but with totals about the same.

In considering items Nos. 4 and 7 it occurs to the writer that the removal of practically all the timber and undergrowth, together with the ground sluicing, hydraulicking and dredging of large areas within the past ten years in both countries, may very probably have been a factor entering into the changes noted.

In conclusion, while there is still abundance of opportunity for research work in the North, it may conservatively be said as regards the problems discussed in this paper, that much progress has been made since they were first outlined by the writer before this Institute in 1913, and that gravels of lower grade are being more profitably mined today than ever before.

In view of the low cost of hydro-electric power at Dawson, it is a matter of regret that the application of electricity to thawing is not further advanced. The experiments for which arrangements had been made, have been temporarily discontinued owing to inability to secure certain equipment during the War. It is to be hoped that at no late day they may be completed. In the meantime reference to the comparative data given above shows, as the result of a season's work on a commercial scale, that thawing by hot water is decidedly more advantageous than by steam, the results being summarized in the expression "four times the ground, in two-thirds the time, with less than half the fuel."

Given, therefore, steam thawing costs and well prospected ground, the margin of lower values which will still permit operation at a profit if thawed with hot water is easily ascertained.

COUNTERCURRENT DECANTATION

By L. B. Eames.

Annual Meeting, Montreal, March, 1917.

In the cyanide process the recovery of dissolved gold from slime pulp, was first accomplished by intermittent decantation. This simple process consists in mixing with the pulp containing the values in solution, a solution of lower gold content, settling the mixture in a tank, then decanting the clear supernatant fluid. The thick pulp remaining in the tank, together with more barren solution, is pumped to a second tank, and again settled and decanted. After several repetitions of this operation values are so far reduced that further washing of the pulp is not profitable. The gold recovery by this process is large, but the plant required is bulky, the labour cost is high, and the amount of solution to be precipitated is very large.

A plant built in the Black Hills of South Dakota, by John Randall, as early as 1901, followed the same principles but attempted to make the process continuous by substituting cone tanks for flat bottomed tanks, the cones operating continuously, receiving a constant feed and discharging a steady stream thickened pulp. The cones were operated in series, the thick underflow of the first one joined with a stream of diluting solution to form the feed to the second cone of the series. Barren solution was added to the tank immediately preceding the discharge tank, and, after being slightly enriched by the low grade pulp in this tank, the solution overflowed to form a diluting solution for the richer feed entering the third tank from the end of the series, and so on back to the richest tank of the series. Clear water was used for the wash in the final tank. This is the principle on which all successful countercurrent decantation plants operate at the present time; but Randall's plant was not successful because of the mechanical difficulties in getting a continuous thick discharge from the conical tanks. A similar plant was also built in South Africa. Here, however the washes were not repeatedly used as in Randall's process, but were precipitated after each contact with the ore. This was also abandoned because of mechanical

difficulties and the cost of precipitating the large quantities of solution necessarily produced. For many years thereafter the continuous decantation was not practiced, and it was not until the introduction of the Dorr thickener that metallurgists began to again consider the continuous decantation principle.

In 1910, two decantation plants were built, the flow sheets of which were similar to that of Randall nine years before, except that Dorr thickeners replaced the cones. One of these was at Mocorito in Sinaloa, Mexico, installed under the direction of C. Dupré Smith; the other was designed by J. V. N. Dorr, assisted by the writer, for the Vulture Mines Co. of Wickenburg, Arizona. While not perfect at first both of these pioneer plants proved so successful as to lead to other installations, which during the past three years, have been numerous.

PART I.—THEORETICAL CONSIDERATIONS.

The following description of the principles and characteristics of the process is submitted; a simple, yet typical, flow sheet, shown in Figure 1, has been selected.

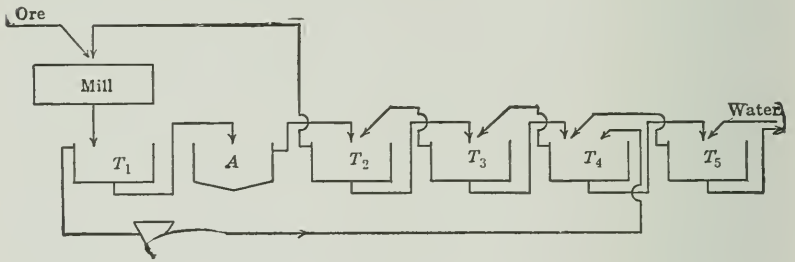


Fig. 1

The flow sheet assumes that crushing is done in solution, the overflow from tank T2 being used for crushing solution. The crushing solution leaving the grinding circuit with the ground pulp, enters T1. The overflow from T1 goes to the precipitation presses. After precipitation of its gold content it is used to dilute the underflow of T3 as it enters T4. The overflow of T5 is also led back to T4. The overflow of T4 mixes with the underflow of T2 to form the feed to T3, and so forth as indicated in the flow sheet. At each succeeding

mixture the solution meets a pulp of higher dissolved content than itself and is enriched while the pulp is correspondingly impoverished. The pulp at each step approaches the discharge end of the mill while the solution goes to the feed end—hence countercurrent decantation.

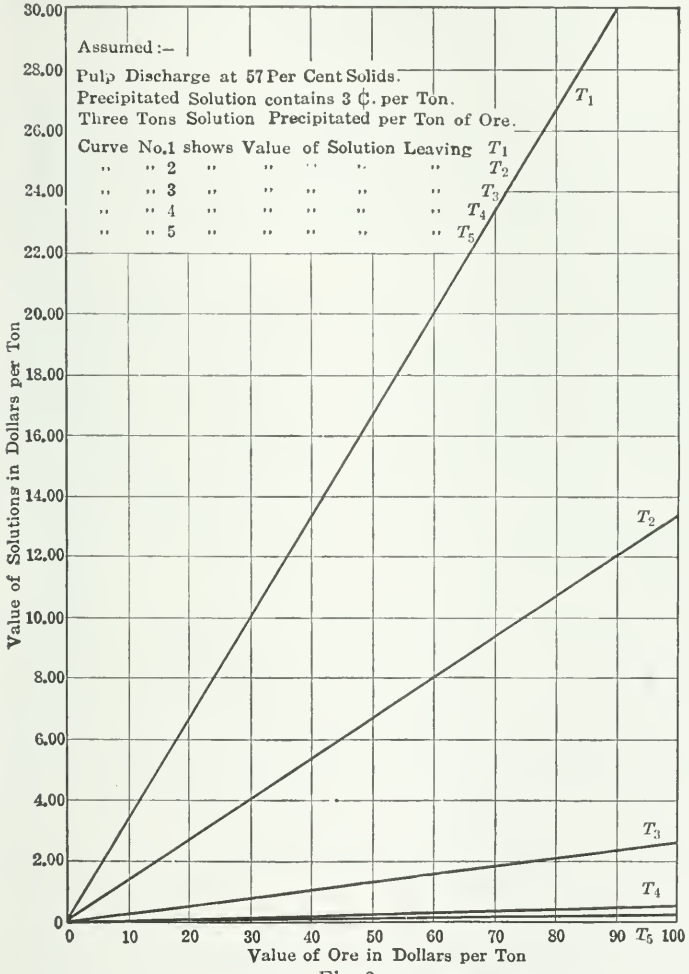


Fig. 2

The principal variables that may affect the efficiency of the process are:—

1.—Grade of ore;

2.—Ratio of solution precipitated to ore treated;

3.—Thickness at which pulp can be discharged;

4.—Cost of chemicals;

5.—Rapidity of dissolution of the gold, and the place in the circuit where it takes place;

6.—Efficiency of precipitation.

Since the decantation process is one involving volumes and dilutions it is possible to calculate accurately what distribution of values should take place under any given set of conditions. As far as possible each one of the above variables has been mathematically considered independently of the rest, and the results have been plotted.

The effect of variations in the grade of ore is shown in Figure 2. and scarcely needs comment. In practice, of course, no such increase as is shown in the gold-solution curve would ever be allowed to take place on account of the difficulty of precipitating such high grade solution and the danger of leakage, but the graph shows conclusively that all solutions increase in value in direct proportion to the increase in the grade of the ore, also that the higher grade solutions increase at a much more rapid rate than the final washes.

The important part played by the ratio of solution precipitated to ore milled is shown in Figure 3. For the particular grade of ore considered, which in this case was $10.00 recoverable per ton, with precipitation to three cents, the economic ratio may be roughly determined by inspection of the lowest curve which represents the value of the solution leaving the last tank of the series with the tailing. It will be noted that the loss in gold increases very rapidly as the amount of solution precipitated is decreased, and that after a certain point the increased recovery, due to increasing the volume precipitated, is very slight.

The final choice of the exact ratio, to be used must be influenced by the cost of precipitation which is mainly affected by the cost of the precipitant. To show clearly the effect of precipitation cost on the economic precipitation ratio, Figure 4 was plotted. I have considered it safe to assume that the cost of increasing the amount of solution precipitated is due to the

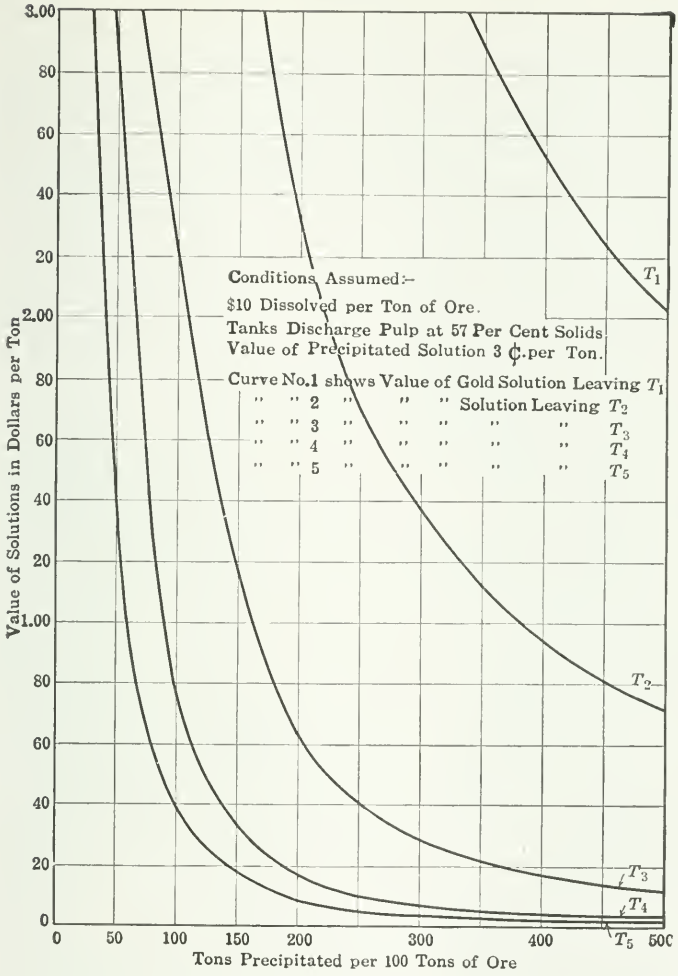

Conditions Assumed:-
$10 Dissolved per Ton of Ore.
Tanks Discharge Pulp at 57 Per Cent Solids
Value of Precipitated Solution 3 ¢. per Ton.
Curve No. 1 shows Value of Gold Solution Leaving T_1
" " 2 " " " Solution Leaving T_2
" " 3 " " " " " T_3
" " 4 " " " " " T_4
" " 5 " " " " " T_5
T_1
T_2
T_3
T_4
T_5
Value of Solutions in Dollars per Ton
Tons Precipitated per 100 Tons of Ore
0 50 100 150 200 300 400 500
3.00 80 60 40 20 2.00 80 60 40 20 1.00 80 60 40 20 0

Fig. 3

Showing effect of variation in the ratio of solution precipitated to ore treated.

additional zinc used. The loss in dissolved gold and the cost of the zinc used in precipitation are both plotted to the same scale in cents per ton of ore. By adding the ordinates of these two curves a third is formed which represents the total loss in gold and zinc. The lowest part of this curve represents the economical range and indicates that, for a ten dollar ore (under the conditions assumed with zinc at 26 cents) from 200 to 250 tons

solution should be precipitated per hundred tons of ore treated. In dotted lines are shown the corresponding curves for a five dollar ore under similar conditions. In this case the range is from 150 to 200 tons. While there is a considerable range within which practically identical results may be expected, it is apparent that each operator should calculate the economic range for the ore to be treated.

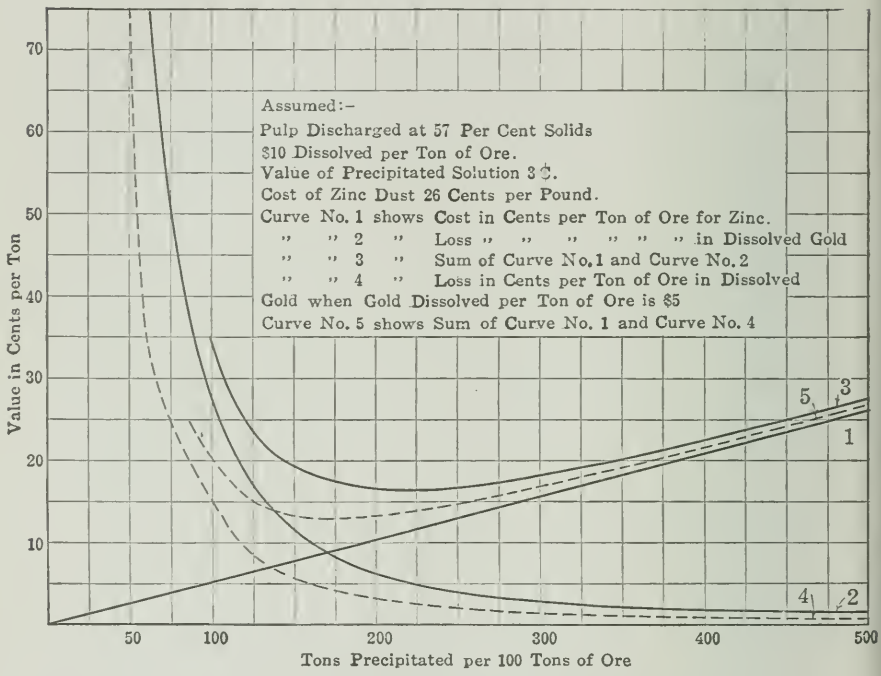

Assumed:-
Pulp Discharged at 57 Per Cent Solids
$10 Dissolved per Ton of Ore.
Value of Precipitated Solution 3¢.
Cost of Zinc Dust 26 Cents per Pound.
Curve No. 1 shows Cost in Cents per Ton of Ore for Zinc.
" " 2 " Loss " " " " " " in Dissolved Gold
" " 3 " Sum of Curve No. 1 and Curve No. 2
" " 4 " Loss in Cents per Ton of Ore in Dissolved
Gold when Gold Dissolved per Ton of Ore is $5
Curve No. 5 shows Sum of Curve No. 1 and Curve No. 4
Value in Cents per Ton
70
60
50
40
30
20
10
50
100
200
300
400
500
Tons Precipitated per 100 Tons of Ore
1
2
3
4
5

Fig. 4

Showing method of determining the proper ratio of solution precipitated to ore treated.

The next, and one of the most important variables to be considered is the thickness, or per cent of solids, to which pulp can be settled. In Figure 5 the values of the various overflows have been plotted to show the effect of variations in the moisture in the underflows. The full line curves represent the values of the overflowing solutions, per ton of solution; but in calculating losses per ton of *ore*, the ratio of the solution to the ore present must be considered. The loss in dissolved gold

per ton of dry ore in the last tank has been plotted therefore in a separate curve (Figure 6), and it is this curve which is especially important in determining the amenability of the ore to decantation. The moisture in the pulp has a direct bearing on the cyanide loss and this also is shown in Figure 5. This has been plotted in pounds instead of in cents, as solution strength and the price of cyanide both affect its value. This curve is numbered 7. The diagram, however indicates that there is a very decided limit to the density of the pulp which can be

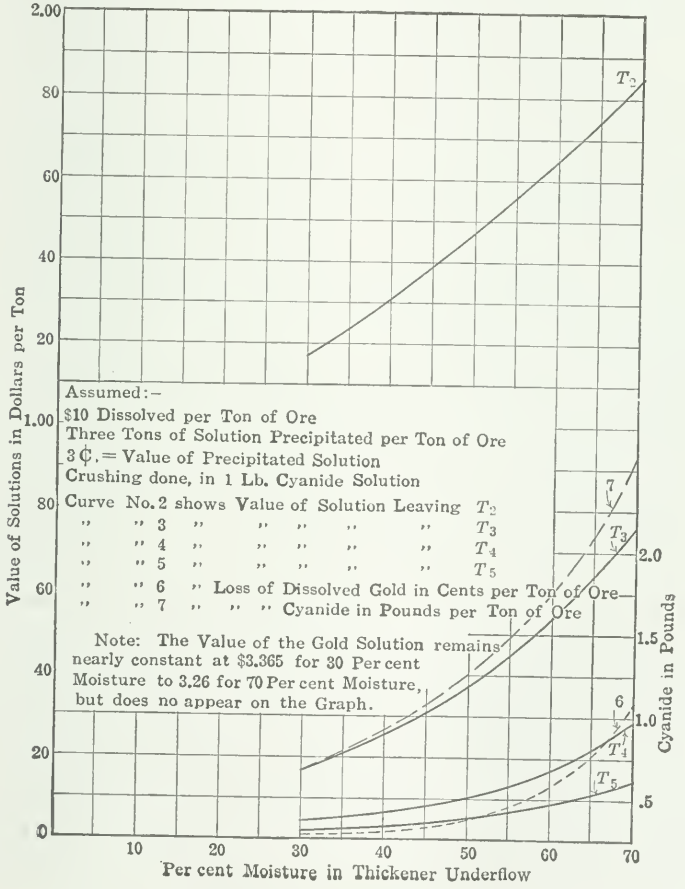


Fig. 5

Showing the effect produced by variations in the per cent of moisture in the underflow of thickeners.

handled economically, and it is clear that the cyanide strength should be kept as low as possible. Operators as a rule seem to be inclined to 'play safe' as regards solution strength, and it is probable that in many cases cyanide could be saved without any considerable loss of gold by using solutions of a lower strength.

In making the calculations upon which the foregoing curves are based, it was assumed that 75% of the gold was dissolved in the grinding department and the remaining 25% in the agitators. Of course this is an assumption which can not be realised, for even with the cleanest ores some gold is dissolved in passing through the tanks that follow the agitators. This is a condition, entering to some extent no matter what method is employed for recovering the dissolved value. Changes of solution during agitation has been the practice for years in the treatment of silver ores, and also in the treatment of concentrate at Goldfield. This is done to reduce the 'lag' of dissolving. I have also found that, on Goldfield ore, pulp from the final agitators, reagitated without change or addition of solution, will yield no more gold; while reagitated filter tailing from the same ore will show a distinct reduction of contained values. The same condition exists in the pulp fed to decantation plants and there is little doubt that some dissolving takes place even in the last tanks of the series. Some of this value is lost, notably the value freed near the final discharge, but much is recovered that would probably not be won by any other method. The superintendent of one of the decantation plants treating Tonopah ore informed me that his sampling indicated that the additional dissolving taking place in his plant was sufficient to off-set his entire dissolved loss and three cents per ton additional. Any considerable dissolving during decantation will be indicated by a difference in the assay value of the solution in the underflow of the tanks as compared with that of the overflowing solution. In practice there is always more gold per ton in the underflow solution of any tank than in the overflow; but in the ores of the Porcupine district the difference is very small. Other causes may, and no doubt do, tend to produce this difference between the overflow and the underflowing solution. Adsorption is probably the most important and perhaps the least understood of these. In the case of the ores of the Porcupine district this phenomenon is of small importance as the ore is

composed of crystalline schists and quartz, and shows little or no tendency to become flocculent under the influence of the solutions used. The gold and silver ores of the western States are in many cases in eruptive rocks, which frequently become flocculent in solution and in so doing seem to entrap a portion of the value in the solution. At any rate there is a much more noticeable difference in the assays of tank effluents in the case of these ores. In some instances this has been attributed to faulty mixing of the products fed to the tank; in most plants riffles, and other devices, are put in launders to ensure thoroughness in mixing, but experience in Porcupine would indicate that the ordinary launder mixes the products satisfactorily. Adsorption then would seem to be the more important cause of these differences, and should be a profitable field for further investigation. Proper precipitation is essential in decantation, as in every other process. It will be observed that the solution leaving T4 is of nearly the same grade as the barren solution and that any increase in the value of the barren solution causes an almost equal increase in the value of the overflow of T4. As the value of the overflow of T5 is one-half of the value of the overflow of T4, one-half of the dissolved gold that reaches T5 in the underflow from T4 is returned in the overflow of T5 and is therefore saved. In other words, substantially half of the value in the barren solution is lost and the other half passes successively through all the tanks again on its way to precipitation. The overflow of all tanks preceeding T4 is increased by precisely the same amount that T4 was increased, as is plainly shown in Fig. 6.

PART 2.—MECHANICAL FEATURES.

The mechanical problems of decantation are simple but are nevertheless worthy of study. The Dorr thickener is universally used for the separation of pulp and solution and is so well known and so standardized that it need only be mentioned in passing. The arrangement of the tanks in plan is largely governed by space available and other local conditions. In respective elevation it is convenient to have the last tank the highest, and the preceeding ones successively lower, so that the overflowing solutions may be transferred or returned by gravity with a minimum of attention. Where such an arrangement is not poss-

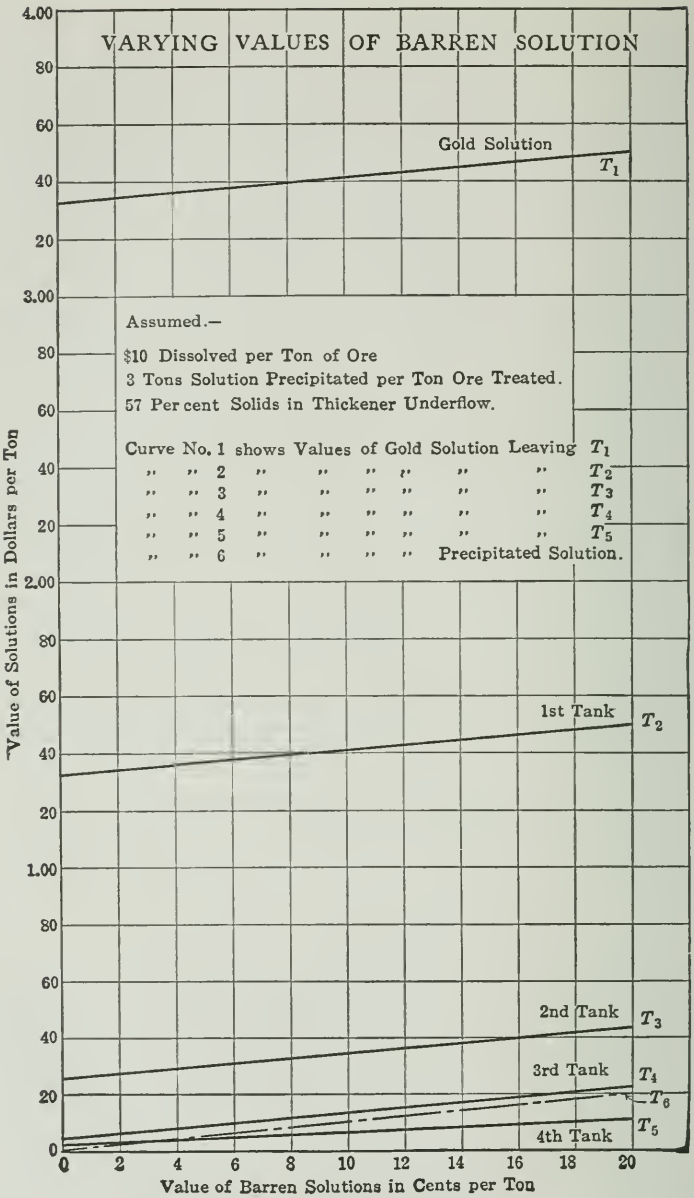


Fig. 6

Showing the effect of variations in the value of barren solution.

ible the tanks may all be placed on the same level, using automatically regulated air lifts to transfer the solutions from tank to tank. As a means of transferring pulp the diaphragm pump is preferred to all others for the reason that it not only pumps efficiently but at the same time measures the volume of pulp transferred; also the flow of pulp is not easily obstructed by foreign matter, such as chips, waste and the like, since there is no contraction of the full bore of the pipe throughout its entire length. The capacity of the pump can be controlled with certainty by means of cone pulleys. As it is operated from the overflow level instead of at the bottom of the tank, the attendant's duties are confined to one floor. The operating cost of a properly designed pump is very low.

Air lifts were used for a time because of their cheapness and simplicity, but they are both wasteful of power and are hard to regulate; the most careful watching will not prevent them from 'running away', that is, transferring too fast and consequently thinning the pulp and therefore sending large quantities of rich underflow solution toward the discharge. If, on the other hand, they are set too slow the gradual thickening of the discharge diminishes the flow and finally stops it.

Spigot discharge to a bucket elevator or centrifugal pump sump has also been tried, the pulp and solution being mixed and elevated together. There are however, several objections to this arrangement, the principal one being the increased cost for more power, and for maintenance of the pumping or elevating machinery. Both solution and pulp must be elevated more than the full height of the tank for each decantation. The small stream of pulp passing through a spigot at relatively high velocity and pressure is subject to frequent stoppages from foreign matter, and even when this does not completely block the spigot it nevertheless interferes with uniformity in operation; in addition two floors have to be served, one below and one above, or near, the top of the tanks.

Some prejudice has been engendered against diaphragm pumps because some of the earlier makes were poorly designed for the work required of them. In addition, defective valves and poor regulation were responsible for much inefficiency. In present practice in most plants in the Porcupine district, cone pulleys

are used for the regulation of capacity, although some operators favour regulation by varying the length of stroke. The valves should be of the floating type, as any hinge device will catch the wood chips that are present in the best screened pulp. The chips lodging in the hinge of the valve cause leaks, which, though small at first cause cutting of the seat and consequent permanent leakage. With the floating valve there is no place for chips to lodge and the whole circumference of the seat is washed by the pulp at every stroke. This type of valve also has the advantage that the lower valve may be placed directly below the upper one and made small enough to be lifted out through the upper valve seat after the upper valve has been removed. As no tools are required for the removal of these valves it is a simple matter to inspect them. The best results have been obtained when the working surfaces of both the valve and seat were of high grade rubber. At first belting was used, but it was found that minute leaks were sure to start, because belting is not yielding enough to close over any chip that may lodge on the valve seat; a leak thus started ruins both valve and seat in the course of a few days. On the other hand valves of rubber, seating on rubber, have been in use six months without the slightest decrease in efficiency and with scarcely perceptible wear. Diaphragm pumps have been operated at speeds varying from 15 to 100, the higher speeds usually in conjunction with a short stroke. The practice at the Hollinger mill has been to use low speed and a stroke as long as the diaphragm can safely stand. Measurements taken on the Hollinger pumps equipped with standard No. 4 Gould diaphragms at 3″ strokes gave results as shown in the following table:

No. of Strokes	Volume per Stroke	Specific Gravity	% of Solids
14.5	.139 cu. ft.	1.54	54.5
23.0	.148 " "	1.48	50.5

Tons Solids Pumped per Day	% Increase in Speed	% Increase in Volume per Stroke	% Increase in Tonnage
76.3	58.5	6.5	50
114.5			

From the above figures, which are typical of numbers of tests made, it may be inferred that the volume pumped is roughly proportional to the speed of the pump, but that leakage is slightly greater on the lower speeds.

By low speeds, and by placing the discharge lips high enough above the discharge valve to allow three or four inches of pulp over the valve at the end of the up stroke, the diaphragm pumps of the Porcupine district are practically free of the splash and dirt that heretofore have been one of the chief objections to them. Where tanks have a settling capacity of over 125 tons of solids per day it may be found advisable to use two diaphragm pumps in parallel, making a duplex, or even a triplex, pump. This arrangement has several advantages; at the lower speed strains of the pump are lessened and distributed; repairs can also be made on one unit without complete stoppage of the tank discharge. To guard against troubles caused by foreign matter, it is advisable to screen the pulp before it goes to the decantation tanks; and where small wood chips are to be expected a fairly fine screen, usually of punched plate, will be of great service in protecting thickeners and pumps.

Every cyanide plant has some method of determining the amount of solution precipitated, and in decantation plants having only one series of tanks the entire tonnage of solution to be precipitated is used at one place. If, however, there are more than one series of tanks it becomes important to split this solution into parts proportional to the tonnage of ore being washed in each series. This cannot be done by regulating the valve at the outlet of the barren line, as trial has shown that under some conditions an increase of as much as 50% can be made in the amount flowing from a pipe line without any visible change in the stream. The difficulty has been overcome by the use of 'V' notch weir boxes, one for each series of tanks. In each weir box is a float compartment and a float operating an indicator on a scale which reads tons per 24 hrs. While the weir box, which can be readily made at any mine, is not a recording instrument and gives only an instantaneous rate reading, its use greatly simplifies the problem of proper distribution of barren solution. Water also should be added in proper quantity and uniformly distributed. A smaller weir box has been found convenient for this purpose.

It is usual to determine the specific gravities of various pump discharges in a decantation plant at least once a shift. In a small plant this consumes little time; but if a large number of measurements are necessary the distance to be covered by the operator is considerable, especially if he has to return to a central point for each weighing. In such plants a portable spring balance, having a dial and revolving hand, is recommended. I have used a milk scale having an adjustable tare-indicating hand which makes one complete revolution for ten pounds. I use a narrow necked can which holds just ten pounds of water when level full. This makes it possible to add a paper dial which can be divided easily so as to read specific gravities directly. The large sample makes accurate readings possible, and enables the operator to determine specific gravity at the place where the sample is taken.

In his paper on the "Dorr Metallurgical Apparatus," Mr. Dorr referred to the future use of tray thickeners, and spoke of the possibility of a complete decantation plant being installed in one tank. While this has not been done as yet, a four step decantation followed by a continuous filter is in operation, the four steps of the decantation being completed in two tanks equipped with single trays. No figures have been made public, but the results are said to be creditable. Another use of the tray in decantation, to be tried shortly at the Hollinger mill, is the use of trays to increase the capacity of an existing plant rather than to decrease the number of tanks installed. Since the same grade of solution is handled in both tank and tray there is no danger of any mixing of solutions of different grades. This plan should decrease the cost of building, tank foundations, and building site, in almost direct proportion to the increase in capacity gained, while tank-cost, power, and labour can be expected to decrease to a marked degree. The outcome of this experiment should have a direct bearing on the future of the decantation process. It must be borne in mind in this connection however, that some slimes require time for their final thickening and consequently necessitate a tank of some depth, while other slimes find their capacity limit in the thin pulp settling rate. For this latter class, as explained by Messrs. Coe and Clevenger in their paper on "Slime Settling," a shallow tank or tray is sufficient,

while for the former class a tank of carefully calculated depth to give the required time for thickening, is required.

The tray thickeners (See Fig. 7) in use at the Hollinger are now operating very successfully, although it has not been advisable to operate them to their fullest capacity. They are now handling daily 70% more than the open tanks and with identical metallurgical results. For a period of three days they were operated at a capacity substantially double that of the simple

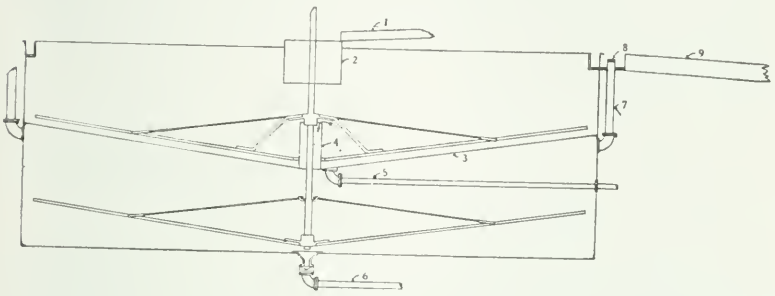


Fig. 7

tank, although the diluting solutions were not quite up to the proper ratio. The connected type tray, as it is called by the manufacturers, has two sets of arms mounted on the same shaft. A steel deck or tray is suspended from the sides of the tank. No braces or beams are used. At the centre is an opening, 18″ in diameter in this case, through which the shaft passes. A tube or sleeve of the same diameter is riveted to the edges of this opening and extends upward a short distance as shown at 4. This tube is open at the top and forms a free passage to the space below the tray. The upper arms sweep the tray in the same manner that the lower arms sweep the bottom, but the discharge opening is necessarily off centre. This is indicated by the connection of the pump suction to the tray, No. 5. The whole feed enters the tank at the center through the launder 1, and the feed well 2. It then spreads out and settles on the tray 3 as in the ordinary tank, the clear solution overflowing the rim as usual. The thickened material is removed through the pump line 5. The heavy feed will soon cause the tank to give indications of muddy

overflow. Solution is now drawn from the bottom section of the sank through overflow lines 7. This is accomplished by lowering the top of the pipe by removing rings 8 so that the head in the tank is greater than that of the pipe. An upward flow through 7 takes place which causes an equal flow down through the tube 4. This flow forms the feed to the bottom section of the tank where it settles and the solids are removed in the usual way through a separate pump line 6 and a separate pump. The result of this method of operating is that the greater part of the solids, especially the coarse, is recovered from the upper section of the tank while a large part of the solution passes through the bottom.

PART III.—THE DECANTATION PLANT OF THE HOLLINGER CONSOLIDATED GOLD MINES LIMITED.

The Hollinger decantation plant consists at present of five rows of 40-feet tanks, four tanks to a row, forming a plant of five units. The tanks are arranged with a difference in elevation of 2 feet 6 inches between steps, the final tanks of the series being the highest so that all solutions gravitate through and out of the plant to preciptiation. The 'Barrett Specification' roof is supported on flat trusses, the lower chords of which pass just above the tank rims. These trusses also serve to support the thickener mechanisms and the walks between the tanks. The diaphragm pumps in this plant were designed by the Company's staff, and have proved very reliable and economical. They are all three-throw or triplex pumps, so that in spite of the large tonnage handled the duty on each diaphragm is light. It is not uncommon for diaphragms to last 300 days, while the life of the present type of valves and seats has yet to be determined. The pumps are used not only to transfer pulp, but also its final discharge. This makes regulation of the final discharge for moisture much easier, more reliable, keeps the work of the operator all on the upper floor and allows the tailing to be discharged at a considerably greater elevation than would otherwise be the case. The barren solution and water added to each row is mea-

sured by separate float-reading weir boxes, assuring uniform results from the various units.

Labour.—The plant is operated by one man per shift who oils all machinery, watches and adjusts the pumps and records their performance. The solution man makes titrations and regulates the addition of water and solution, but has no other duties in the decantation plant. A repair man on day shift makes all repairs and has time for other work.

Power.—The power including motor and line shaft losses for the twenty tanks, four of them containing tray mechanisms, is 9.2 h.p. while the corresponding figure for the 24 three throw diagram pumps in the plant is 11.1 h.p. making the total power consumed in the plant 20.3 h.p.

Cost.—The costs for the twelve weeks from January 28th. to April 21st., 1916, have been taken as typical of what is accomplished in this plant at its present capacity. During this time 85,854 tons were decanted at a cost of $599.00 for supplies including power, and $1,194.00 for labour: of .007c. per ton for power and supplies, and .0139 for labour, making a total of .0209 cents per ton for decantation. Labour costs, no doubt are higher now than they will be in the future, as a greatly increased tonnage is to be treated; while supplies and power costs should remain nearly the same. The present cost is about 40% of the cost of filtering the same daily tonnage with leaf filters.

Extraction and Recovery.—From the ores of the Porcupine district the recovery by dilution seems to be almost the theoretical maximum. Adsorption does not seem to have any appreciable effect. There is a slight dissolving during decantation, which, while it adds to the recovery, makes the soluble loss somewhat greater than it would otherwise be. The figures quoted below on chemical consumption and recovery refer to only two units of the Hollinger plant. The figures for these units are given because the other units of the mill share their feed and likewise their barren solution with the original Moore filter plant, but for commercial reasons the two units in question have been given

a separate solution system and separate precipitation presses. These two units are therefore the only ones for which all the figures are available. In comparing the results quoted, however, it should be borne in mind that the flow sheet has been modified in this plant because of limitations of space, so that the overflow of T2, instead of that of T1, goes to precipitation. The effect of this is to raise the theoretical value of the overflow of the last tank 3 cents at 3 to 1 precipitation.

A statement of results follows:—

Period covered:—Same as that for which costs were given—from January 28th, 1916, to April 21st. 1916.

Tons of Ore treated:—38,885.

Value per ton of ore treated:—$8.92.

Ratio of ore to solution precipitated:—100 to 285.

Tons solution precipitated:—110,604.

Strength of cyanide used:—0.9 pounds per ton, or .0045%.

Cyanide added per ton of ore:—.46 pounds.

Difference between pulp feed and pulp discharge for first tank after agitators:—25c.

Average moisture in tails:—45%.

Average value of barren solution:—3.2c.

Dissolved gold per ton of solution discharged:—11.71c.

Dissolved gold per ton of ore discharged:—9.57c.

Taking into consideration the flow sheet, the grade of ore treated, the barren solution used, and the thickness of pulp attained, it is theoretically possible to reduce the overflow of the last tank to 7.6c., leaving a difference of 4.1c. to be accounted for by continued dissolving, adsorption, etc. Viewed in one way it may be said that actual losses are 54% higher than theoretical, but where one is dealing with samples so easily affected by faulty manipulation, and where any error, except losses in assaying, tends to raise the results, a check to 4 cents appears satisfactory. The average loss would have been somewhat less if the occasional high results had been omitted: but this was not done.

From the foregoing I believe that one is warranted in concluding that a reasonably accurate forecast of the results to be expected from a decantation plant can be made, and that the results will compare very favourable with the results obtained from filter plants.

In conclusion I acknowledge my indebtedness to Mr. P. A. Robbins, Managing Director of the Hollinger Company, for permission to quote results from the Hollinger plant.

REFRACTORY MATERIALS IN CANADA.[1]

By N. B. Davis.

Annual Meeting, Montreal, March, 1917

The most important accessory to the practice of metallurgy and ceramics is a suitable refractory material—a substance that will withstand high temperatures and the accompanying physical and chemical conditions of abrasion and corrosion.

Blast furnaces, melting furnaces of every description, converters, etc., (the tools of the metallurgist), kilns used in the lime, cement, glass, and clay industries, (the tools of the ceramist) all require to be lined with a refractory material, either acid, basic, or neutral in character, depending on the nature of the work.

In Canada the great majority of the industries using refractories are dependent on American and European importations, a condition that has been a natural outcome of the transplanting of industries, of a small population and of a long frontier. There has also been a lack of knowledge of refractory raw materials in the older parts of the Dominion. It has been but recently, under the stress of war conditions that our industries using refractories have begun to appreciate the necessity of utilizing Canadian raw materials for the manufacture of the finished article into which they enter.

The fuel supply is a very important item in the manufacture of refractories, since high temperatures must be used in burning the ware. Ontario, the principal manufacturing province, has not a local supply of coal, and hence imports both fuel and refractories, largely from Pennsylvania.

Quebec imports all its coal supply from Pennsylvania and Nova Scotia. Refractories are drawn from Pennsylvania and Scotland. There is a small local production of special shapes manufactured from imported American clays.

The Maritime Provinces draw coal from Nova Scotia fields and refractories mainly from Scotland. The Nova Scotia Steel

[1]Published by permission of the Director of the Mines Branch, Canadian Department of Mines.

& Coal Co., has recently installed a firebrick plant at Sydney Mines and started the manufacture of fireclay shapes for use in ladle linings, generative chambers, flues, etc.

The industries in the Middle West—Manitoba, Saskatchewan, and Alberta—draw coal from the United States and the Alberta fields. Refractories are largely imported from St. Louis district and from Pennsylvania. In the Province of Saskatchewan there is one plant in operation making firebrick at Claybank, about 25 miles south of Moosejaw.

The Pacific coast industries use local coal and import some refractory materials from the United States and Scotland. The most extensive plant manufacturing refractory goods in Canada is located at Clayburn, British Columbia. It includes the manufacture of retorts in addition to standard firebrick and special shapes for furnaces.

The manufacture of refractory ware in Canada is therefore limited to a small plant at Sydney, Nova Scotia, one at Montreal, and one at St. John, Quebec, one fair sized plant at Claybank, Saskatchewan, and a larger plant at Clayburn, British Columbia.

Distribution of Raw Materials.—Neutral Refractories. Clays and Shales.

Maritime Provinces.—The clay and shale beds associated with the coal seams in Nova Scotia are for the most part easily fusible materials, which cannot be classed even as semi-refractory.

The only materials, so far found in the coal districts, which approach the requirements for refractoriness, consist of a 3-foot bed of plastic clay overlying the 13-foot seam at Inverness and a 4-foot bed of hard shale underlying the No. 3 coal seam at the mine of the Intercolonial Coal Company at Westville. These have a refractory value up to 2600°F.

The most refractory clay known in Nova Scotia does not occur in the coal measures, but is found in unconsolidated Cretaceous clays at Shubenacadie, on the line of the Intercolonial Railway. Tests on this material show it to be a number 2 refractory, deforming at cone 30 (3100 degrees F.). The white or

grey clay at Middle Musquodoboit, 16 miles east of Shubenacadie, is of similar age and character.

A felsite rock occurring at Coxheath near Sydney has refractory qualities but is non-plastic. When crushed and bonded with plastic fireclay it can be manufactured into a very desirable firebrick.

The most refractory clay so far found in New Brunswick occurs under the thin coal seam at Flower Cove, in the Grand Lake coal area. It is only semi-refractory, as it deforms at the softening point of cone 25 (2966 degrees F.)

Quebec.—The only important source of refractory clay in Quebec is confined at present to the kaolin deposit at St. Remi, d'Amherst, situated 70 miles northwest of Montreal. Preliminary experiments with the crude kaolin for the manufacture of firebrick have given promising results and the material is now being tested on a commercial scale. A large quantity of this material is being revealed in the development work at the mine and it is probable that intensive prospecting of the adjacent ground will result in the finding of other bodies of kaolin.

An occurrence of residual clay is reported in Two Mountains county, near the line of the Canadian Northern Railway. This material has refractory qualities, but the deposit is considered too small to be of economic value.

Ontario.—Fireclay is of rare occurrence in the Province of Ontario, and none is found in the more settled portions. There is a heavy annual importation of refractory goods to supply the needs of the large and varied industries. A preliminary survey of the province has resulted in finding only two localities in which fireclay occurs; one being at the Helen iron mine in Michipicoten, where a diabase dike has weathered into a residual clay. This material softens at about cone 26 (2950 degrees F.) The other deposit occurs on the Missanaibi river about 40 miles north of the crossing of the National Transcontinental railway. It appears to be extensive, but owing to the remoteness cannot be said to be of commercial importance at present. The search for refractory materials in Northern Ontario has never been seriously undertaken but the prospects of finding new deposits are good.

Manitoba.—No good refractory clays are reported from Manitoba, but some prospecting for this purpose remains to be done, particularly in the Dakota horizon of the Cretaceous. The Odanah horizon of the Cretaceous outcropping at Turtle Mountain, La Riviere, and in the Assiniboine valley near Virden, consists of a hard grey shale that may be regarded as semi-refractory. It withstands temperatures up to cone 15 (2600 degrees F.) before deforming. Like the semi-refractory shale used at Sydney, N.S., it may have a local value as a refractory for medium temperature work. This shale is known as far west as Tantallon in the Qu'Appelle valley.

Saskatchewan.—The southern part of the Province of Saskatchewan is particularly rich in refractory and semi-refractory clays. For a number of years a drypress face brick plant has been in operation at Claybank in the Dirt Hills south of Moosejaw. The material used is a high-grade (No. 2) fireclay and although the most of the brick manufactured in the past have been sold as face brick, a certain amount have been used as firebrick. During the past year the company has been reorganized and it is proposed to manufacture standard firebrick and special shapes as well as the regular line of face brick.

Similar clays to those used at Claybank occur at the north end of Lake-of-the-Rivers, near Mitchellton on the Canadian Northern; also along the Canadian Pacific, Sterling branch at Willows; south of Twelve Mile lake; and along the Frenchman river valley in the Cypress Hills from Eastend to Palisade. The clays of the Lake-of-the-Rivers valley, Wood Mountain and Dirt Hills, are more refractory than the clays further west. The former deform around 3150 degrees F. while the latter seldom remain intact above 2900 degrees F. The clays of the Cypress Hills are more suitable for the manufacture of vitrified clay ware, such as sewerpipe and stoneware.

Alberta.—No fireclays have been found among the clay and shale deposits of southern Alberta up to the present. Beds of high-grade clays occur in northern Alberta along the Athabasca river for some distance north of McMurray. Most of these clays are of the stoneware type and semi-refractory, but one bed was found to meet the requirements of a fireclay. These clays are

apparently in the Dakota formation at the base of the Cretaceous, and intimately associated with the tar sands, in fact some of them are rendered almost useless by the impregnation of finely divided carbonaceous matter.

British Columbia.—The most important sources of refractory clays at present known in British Columbia are the Tertiary beds in Sumas mountain, where about 15 feet of hard fireclay is interbedded with other shales of a semi-refractory character, together with some useful vitrifying shales. This section on the whole contains the best series of materials known in Canada for the manufacture of a varied range of clay products.

Refractory shales with similar associations to those at Sumas Mountain occur at Blue Mountain, near Whonnock on the Canadian Pacific Railway. Fireclay of residual origin occurring at Kyuquot is shipped to Victoria for the manufacture of stove linings and sewerpipe.

Several samples of so-called kaolin have been forwarded from British Columbia to the Mines Branch for examination, but all of these have proved on testing to be either diatomaceous earth or volcanic ash. It is quite probable, however, that kaolin deposits of commercial value will be discovered in the Interior Plateau region.

CHROMITE

Of the neutral refractories, chrome brick are of the greatest importance in the metallurgy of iron, copper, antimony and tin. In the manufacture of these bricks, chromite, a sesqui-oxide of chromium (FeO, Cr_2O_3) usually termed chrome ore is used. It is extremely refractory and practically neutral, which properties give it its value. The natural chrome ore is mixed with a suitable binder, when necessary, such as fireclay or a mixture of low calcined MgO and $MgCl_2$, and manufactured into brick. The chief use of chromite brick is as a buffer or separating course or lining between a basic bottom or hearth and acid or neutral wall and crown in order to prevent any reaction at high temperatures between the basic and acid materials.

The Province of Quebec, as in the case of magnesite, has

become a large shipper of chromite, the production being 14,397 tons in 1915. Most of the production comes from the Black Lake area, where it occurs as irregular masses and disseminated grains in serpentine rocks.

For refractory purposes chromite ore is sold on the market on the basis of a minimum of 25% Cr_2O_3. The material is hand cobbed and also wet concentrated for the market.

Chromite is known to occur at Tulameen and on Scottie creek in the Clinton Mining Division, and on Taylor creek in the Lillooet Mining Division, British Columbia.

The geological associations of the chromite and magnesite in these areas are upturned Cretaceous sediments, intruded by peridotite, being similar to the occurrence of the same minerals in California.

Bone Ash

Bone ash is a less prominent neutral refractory but has its use in the metallurgy of lead and for assaying.

The amount of bone ash used in Canada for this purpose is limited and the market for a susbtitute is small. Experiments are projected to ascertain the value of the mineral apatite as a substitute.

Acid Refractories

Two kinds of silica refractories are used in the various industries, one having lime as the bonding material and the other clay.

For making silica brick, pure quartzite is crushed and milled in a wet pan with milk of lime. The milling is done as thoroughly as possible in order that each particle of quartzite may be coated with lime and a slightly cohesive mass produced. The bricks are made by hand and partially dried, then repressed by machinery and sent to the kiln for burning.

Some hard sandstones in the coal measures in England contain enough clay in their composition to form a bond for the quartz grains when the rock is crushed and milled with a little

water. This when made up into shape and burned is known as Gannister brick. A later development consists in grinding pure quartzite with plastic fireclay for the manufacture of this type of refractory.

Ordinary quartz begins to swell at a temperature of 850° C. and continues to swell the higher the temperature rises, until most of it is converted into cristobalite and tridymite. If the burning of silica bricks is not carried to a high enough temperature in the manufacture they will expand on reheating, sufficient to cause serious damage to the furnaces into which they are built.

Some trials with various occurrences of quartz when bonded with lime and clay were made in the Mines Branch laboratories.

Quartz from the feldspar mines near Verona, Ontario, was bonded with 2% caustic lime and 5% plastic fireclay, and burned to cone 20 (2780° F). In both cases the quartz showed considerable expansion and destroyed the bond produced in firing, making the bricks weak and crumbly.

Quartz from a large mass in Levant township, Ontario, was also tested in a similar manner and it likewise resulted in a swollen punky body.

Quartzite from Killarney, Ontario, bonded with 3% caustic lime and with 5% kaolin produced a good, strong body with both bonds and only a moderate amount of swelling.

The quartzite wall rock in the kaolin mine at St. Remi, Quebec, is impregnated with specks and streaks of kaolin for a distance of 130 feet from the main kaolin body. An average sample of the rock contained about 11% of clay. A sample of this material was crushed and milled, made up into trial pieces, burned to cone 9 (2390° F.) in a gas kiln and afterwards to cone 18 (2714° F.) in an electric kiln. The test pieces were hard and dense, showing that fused bond had been effected between the quartz grains and the kaolin. The body was strong and showed no undue swelling on the part of the quartz. This material appears to be suitable for the manufacture of an acid refractory brick of the gannister type, as it showed no indication of softening at a temperature of 2900° F.

Quartzites of Pre-Cambrian age like those occurring at

Killarney and St. Remi are also found conveniently situated at other points in Eastern Canada.

The Potsdam sandstone at the base of the Paleozoic in the St. Lawrence and Ottawa valleys is often a quartzite, but tests made on this material for the manufacture of refractory brick were not encouraging, as too much fine grained material results from crushing, and the iron content is too high. The Potsdam sandstone, however, is used as a glass sand in places where it is free from iron as at Beauharnois, Quebec.

Practically the same results were obtained in tests for silica brick when using the Orikany sandstones in the Devonian rocks of Southwestern Ontario. This sandstone is crushed and washed for glass sand at Nelles Corners.

A considerable quantity of white quartzite is contained in the Summit series of Lower Cambrian age in the Kootenay district, British Columbia. These quartzites outcrop on Kootenay lake at Crawford bay, and are probably the purest rocks of this kind in the province.

No tests were made of this material, but the character of the rock indicates that it might be used in the manufacture of silica brick or as a source of silica for other purposes.

The tests so far made show that all forms of quartz are not suitable for the manufacture of refractory ware. Quartzites give much better results than igneous quartz or sandstones. The sharp splintery particles which quartzite yields on crushing, form an interlocking bond which is essential to strength in the finished brick. Brick made from the rounded grains of sandstone lack the proper strength for handling.

Although vein quartz breaks down into splintery particles on crushing it is not a desirable material to use on account of its behaviour on heating.

Basic Refractories

Basic refractories include magnesite, dolomite, and bauxite, or the artificial product alundum, the first named being the most important. Until two years ago the production of these materials in Canada was very small; 340 tons of magnesite were exported in

1912. With the coming of the war and the consequent dislocation of trade, the American manufacturers of basic refractories hurriedly sought an American supply. Attention was turned to the occurrence of magnesian dolomite in the Grenville rocks of Quebec and prospecting in the vicinity soon developed the fact that there was considerable dolomite magnesite available. The rock is sufficiently high in magnesite to enable the shippers to meet a requirement of approximately 85% $MgCO_3$. The production has grown rapidly as shown by the following table, supplied by the Statistics Division of the Mines Branch.

Production of Magnesite in Canada.

	Tons	Value
1908	120	$ 840.00
1909	330	2,508.00
1910	323	2,160.00
1911	991	5,531.00
1912	1,714	9,645.00
1913	515	3,335.00
1914	358	2,240.00
1915	14,779	126,535.00
1916	55,413[1]	563,829.00

Lining the bottoms of steel furnaces is the principal use to which the Grenville magnesite is applied.

For this purpose the magnesite is calcined, then broken into small fragments and mixed with from 15% to 40% of crushed basic iron slag. It is said to give as good results as the magnesite hitherto imported from Austria.

None of the Grenville magnesite appears to be used in the manufacture of refractory brick, on account of its high lime content.

Experiments are now in progress in the Mines Branch laboratories by Mr. Frechette with the object of reducing the lime content by mechanical means. The process of separating the lime from the magnesite is based on the difference in character

[1]Includes 635 tons marketed from Atlin, B.C.

between these materials after slaking the calcined mass. If this separation can be carried out in practice a supply of high-grade Canadian magnesite can be placed on the market.

A considerable quantity of high-grade magnesite associated with serpentinized peridotite was found by Mr. Drysdale in the Bridge river district of the Lillooet Mining Division, British Columbia, but these deposits are situated rather too far from railway transportation at present.

The hydro-magnesite deposits at Atlin, in the northern part of British Columbia, vary from the above mentioned occurrences as they are superficial deposits in a finely divided condition and not rock masses. This material although remotely situated could be used to advantage on the Pacific coast in the chemical industry and for the manufacture of special cements.

Where refractory bricks containing a high percentage of alumina and a low silica content are required, bauxite is the material employed in their manufacture. No record has so far been obtained of the occurrence of bauxite in Canada.

Discussion

Professor M. L. Baker:—Mr. Davis' paper on these refractory clays, has proven very interesting to me in two or three respects. One is the disproving of the old idea that we have no residual clays in Canada. There is evidently one place in Quebec and possibly two in Ontario in which these clays have been found. I think it would be interesting to know from him, what that particular deposit on the Missanaibi River was formed from. The reason I mentioned this is that so much of Canada is occupied by similar pre-Cambrian rocks that if we knew the origin of this particular clay, we could then be on the watch for similar deposits in other parts of the Dominion, and as these residual clays are so scarce in Canada and, therefore, so valuable, it would prove of considerable interest to know their origin.

Another point in connection with this paper, of much interest, was that in reference to the manufacture of chromite brick. Within

the last two or three years large areas of peridotite have been found in northern Ontario. Deposits of nickel are found associated with them, and they are almost always chrome-bearing, not in the sense that they would supply good chrome iron ore, but they almost all carry a small percentage of uniformly disseminated chromite. Peridotites weather so easily that clays derivable from these northern peridotites might prove of value in this chromite brick manufacture.

MR. N. B. DAVIS:—So far the Missanaibi river clays referred to have not been examined in detail; but all information to hand indicates that they are sedimentary in character, and probably belong to Tertiary deposits of that region. The material probably was originally derived from the weathering of Pre-Cambrian rocks.

The material resulting from the normal weathering of peridotite rocks would be too high in ferruginous clay to make it of value as a refractory.

It would be well for prospectors to be on the watch for white clay when prospecting in the Pre-Cambrian areas of Quebec, Ontario, and Manitoba.

It should be noted that the Ceramic Division of the Mines Branch is prepared to examine material free of charge.

THE DEVELOPMENT OF CANADIAN MAGNESITE

By Harold J. Roast, F.C.S.

Annual Meeting, Montreal, March, 1917

Magnesite is essentially carbonate of magnesium ($MgCO_3$). The most objectionable impurity associated with it is carbonate of lime. For some purposes the presence of iron oxide is a detriment, but for others it is considered desirable rather than otherwise. Hydro-magnesite is magnesite associated with hydrate of magnesium and water ($3MgCO_3.Mg(OH)_2 + 3H_2O$).

Magnesite commonly ranges from white to greenish white in colour, varying in the case of the crystaline variety, while the amorphous form is usually yellowish white. Its specific gravity is about 3; hardness 4; and it contains, when pure, 52.4% carbon di-oxide and 47.6% magnesia. It takes a high polish whether amorphous or crystaline.

The two best known deposits are those of Austro-Hungary and of Greece. Austrian magnesite varies from fine yellowish white to coarse bluish white crystals. Its composition is:

Silica and Alumina	1.0%
Oxide of Iron	3.0%
Lime	1.0%
Magnesia	45.0%
Carbon di-oxide	50.0%
	100.0%

Its composition dead burned is:

Silica	2.5%
Alumina	0.5%
Iron Oxide	7.0%
Lime	2.75%
Magnesia	86.5%
Carbon di-oxide, etc.	0.75%

Its distinguishing feature is its high iron content; which assists in making it sinter when formed into refractory brick. Most of it is calcined to a degree that not only drives off all carbonic acid, but actually shrinks the magnesia to a dense dark coloured mass.

Grecian magnesite is of next importance to Austrian, the chief deposits now worked being those on the Island of Euboea. It is conchoidal in fracture (like the fracture of broken china) and dead white in colour. Its analysis is:

Silica	3.0%
Iron and Alumina	.5%
Lime	2.0%
Magnesia	44.0%
Carbonic Acid	50.0%
Water, etc	0.5%
	100.0%

As will be noted, it contains very little iron. The material formerly mined contained only a trace of lime, but of late years its lime content has been as high as 2%.

Magnesite occurs also in Germany near Frankenstein, Silesia, but the deposits have not been developed as extensively as the Austrian. It is very low in iron and lime. Italy has deposits in the Turin district and on the Island of Elba. The silica content is high, about 8%. Large deposits occur in Macedonia. They are similar in nature to those of Greece, but are not worked to any extent at present. Norway produces magnesite somewhat similar to that of Austria. It is burned and manufactured into refractory bricks containing about 9% of silica, 5% oxide of iron, and 84% magnesia. In Russia it occurs in the Urals. Many thousands of tons have been mined, but Russian magnesite is not supplied to the American market. In Africa in the Transvaal, magnesite of good quality, similar to the Grecian has been found, but contains even less lime. A deposit of good quality is worked in a locality in India. Here a calcining plant has been operated, and bricks manufactured. Large deposits have been found in

Australia, but so far no serious attempt to develop them has been made.

On the American continent, magnesite is mined in Venezuela, and some has been marketed in New York. It is similar to the Grecian ore. A large boulder deposit exists on the Island of Margarita, Mexico. Some magnesite from this source has also been marketed in New York, but the supply has been very irregular on account of the inadequacy of the ocean transport service, Magnesite in the form of small water-worn pieces, is found on a part of the coast of the Island of Cuba, hence it is probable that a deposit exists somewhere on the Island. The largest deposits in America are worked in California. Their development has been proceeding for some years past. The magnesite is similar to the Grecian, and like it has not been found as satisfactory as the Austrian product for the manufacture of refractory brick. Before the war nearly all the calcined magnesite produced went to the local Californian paper mills. Since 1915, more capital has been made available for the working of the deposits, and branch railways have been constructed to the mines, thereby enabling the ore to be loaded directly on cars. Formerly the mines were twenty miles from a railway. The magnesite has recently been sold to manufacturers of refractory material, and this trade has largely exceeded that of the paper mills. Furthermore ore similar to Austrian magnesite, having 6.44% of iron and 1% of silica with no lime, has been discovered in California. The whole of the product so far developed has been sold to local steel companies for the linings of furnaces and other refractory purposes. In one of the Californian mines a 60-foot rotary kiln has been erected. From the foregoing, it will be apparent that magnesite is very widely distributed.

CANADIAN MAGNESITE

An average analysis of Canadian magnesite is:

Silicia, Iron and Alumina	3%
Lime	9%
Magnesia	40%
Loss on ignition	48%
	100%

The principal deposits of magnesite in Canada are in Argenteuil County, Quebec, between Calumet and Grenville, about twelve to fifteen miles north of the Ottawa River. Three companies are operating there, although as yet the greater part of the development has been made by two of them.

The roads to the deposits are rather poorly maintained and hilly. The quarries are some 700 feet above the level of the Ottawa River. The district is well wooded, and at present for calcining, wood is the fuel used. The deposits are uniform in many respects. The magnesite outcrops in the form either of small bluffs or of mounds, commonly in small swamps. It is characteristic of magnesite that where exposed to the weather, it is often altered to a hydro-magnesite, of bright white colour, which fact is of assistance to the prospector. Where the deposits are being worked a solid white crystaline magnesite extends to a depth of eighty feet or more. Beneath this is silicate of magnesia, and talc mixed with mica and serpentine. It is not in veins but rather in masses, bounded by dolomite, with streaks of soft pink calcite here and there, and occasional intrusions of diorite.

The similarity between dolomite and magnesite is such that from appearence only, none, not even those intimately acquainted with the rock, can decide as to whether the lime content will be 4% or 44%. This fact has led, and still leads, to much disappointment on the part of prospectors and others interested in the many samples of white rock brought to Montreal from time to time. The method used to win the magnesite is open quarrying rather than mining.

MANUFACTURED PRODUCTS

In the earliest development of the deposits, the rock was shipped in the crude form for the purpose of making liquid carbonic acid gas. To produce this gas from magnesite the rock was first crushed to a size of about 3-inch ring and fed into a suitable retort furnace, the retorts being heated in an oven so that the rock is heated indirectly. A good red heat is considered the best temperature; not that this heat is required for the dissociation of the magnesite, but because it gives the best com-

Quarry and picking platform of the Scottish-Canadian Magnesite Co., Grenville tp., Que.

Quarry No. 2 of the North American Magnesite Co., Grenville tp., Que.

mercial results, having regard to time and the pressure at which the gas is released. As a matter of fact, magnesite commences to dissociate at a temperature of 509° F. and the whole of the carbonic acid gas is given off a temperature of 950° F. Limestone on the other hand requires 1517°F. to complete the decomposition. The retorts are closed gas tight, and from the rear end suitable pipe connections lead the gas from all retorts into a main gas line. At this stage the gas is 99.9% carbonic acid. It has, however, a disagreeable odor which is removed by passing the gas through potassium permanganate purifiers. The gas is then led into a gasometer from which it is compressed in stages to a final pressure of about 1500 lb. per square inch at a temperature of about 60° F. and so liquified. The chief use of this gas is in the manufacture of soft drinks. In Europe, especially in Germany, it has a larger field of usefulness. Other forms of furnaces are also used, especially in California; but modern methods are opposed to the continued use of magnesite for the production of carbonic acid gas except in very special cases.

The carbonic acid gas made in Canada previous to the use of Canadian magnesite had been derived from Greece, but owing to strikes in the Grecian mines it became increasingly difficult to get deliveries. The manufacture of carbonic acid gas, as outlined, produced, as a by-product, calcined magnesite. The chief outlet for the sale of this product was in the United States for use in the manufacture of composition flooring. On using Canadian magnesite, as described above, it was found that the calcined product was objected to on account of the lime content. This caused considerable experimental work to be undertaken with the result that it was found feasible to so calcine the Canadian magnesite that the calcined product made an artificial floor which was quite the equal of that produced from the Grecian article. The essential feature in the calcination of the Canadian magnesite for flooring purposes is the calcining of all the carbonate of magnesia to oxide, leaving at the same time all the carbonate of lime untouched. This was found to be possible in retorts, which were a modification of those used in the production of carbonic acid gas. The flooring trade requires very finely pulverized magnesite, and to obtain this the calcined product is crushed and fed into sets of vertical burr stones so adjusted that a product giving 99%

through a 100 mesh screen is produced. A typical analysis of this material is:

Silicia, Iron and Alumina	5%
Calcium carbonate	23%
Magnesium carbonate	13%
Magnesium oxide, or magnesia	59%
	100%

It will be seen from these figures that the carbonate of lime is inert while nearly all the carbonate of magnesia is converted into oxide, which is the condition needed in regard to the formation of the cement required. In the manufacture of composition flooring, this calcined magnesite is mixed with some fibrous filler such as wood flour or sawdust and this again with a solution of choloride of magnesium, the resultant pasty mass is then spread on the foundation and carefully trowelled. In twelve hours it is fairly hard, and in twenty-four hours will stand walking on, though it does not reach its maximum hardness for several days, depending on the weather. The tensile strength of this flooring is 600 lb. per square inch at the end of four days.

By the time the flooring trade in Canadian magnesite had been initiated, the Canadian deposits had been better developed, and it was desired to extend the manufacture and sale of magnesite products. With this end in view the crude rock was calcined in an upright kiln similar to the modern lime kiln. This consists of a cylindrical shell of sheet iron lined with suitable fire brick. At the bottom of the cylindrical form, the kiln runs down to an inverted cone; at the juncture of these two parts are the fire boxes so arranged that the flame from the fuel impinges on the rock, the heat working its way through the rock and being thus gradually absorbed. In this way the rock going in at the top is gradually heated until it comes into direct contact with the flame when the heat reaches the maximum and all the carbonic acid gas is driven off. The calcined material then works its way down into the cooling cone, from which it is drawn off at regular intervals. The process is continuous. The calcined magnesite is in the form

of fairly large lumps which are easily broken up. The analysis of this material is about:

Silica, Iron and Alumina	6%
Magnesia	75%
Carbonate of Lime	4%
Lime	15%
	100%

Part of this calcined material was shipped unground to certain pulp mills for use in the digestion of wood pulp. The magnesia combines with the sulphurous acid forming magnesium bi-sulphite. It is found that the magnesium bi-sulphite is more stable than the corresponding lime salt and that it has a more satisfactory solvent action on the free resins of the wood. A portion of the material was ground to pass through 100 mesh screen, and shipped to those pulp mills which preferred the ground product. It was found that the volume of the trade along these lines was not great enough to make it profitable, on the narrow margin of profit on which producers had to work, in view of the competition of ordinary lime used to a large extent by the pulp mills.

There remained the two largest uses of magnesite, viz.: refractory linings and hearths for steel furnaces, and the manufacture of magnesite brick. In the introduction of Canadian magnesite for these purposes, the chief difficulties were the presence of lime and the absence of iron. At first the difference between Canadian and Austrian magnesite in physical appearance owing to this absence of iron was the main objection to even trying the new article. In this connection the following results of experiments made by the writer may be of interest. Canadian magnesite will not fuse or slag below a temperature of 1800° C. It will not shrink or assume a dense form until a temperature of about 1700° C. is reached. On the other hand, it will become dense and have the usual properties of ordinary dead burned magnesite after being heated to that temperature. It has been stated that lime in magnesite will cause the dead burned article to absorb moisture and so slack to a harmful extent. A series of tests showed that Canadian dead burned magnesite carrying as much as 16%

of lime does *not* slack to any extent, even after very fine grinding and prolonged exposure to moist air. The dead burned Canadian magnesite is dead white, and is very hard being capable of scratching glass. The best dead burned magnesite that could be obtained at the time of these experiments absorbed from .5% to .75% after exposure for three days, during which time the Canadian article, as already stated, showed no appreciable increase in weight.

This was the stage at which the development of Canadian magnesite had arrived prior to the outbreak of war. The exclusion of Austrian and Grecian magnesite, and the consequent scarcity, resulted in a trial, or rather a series of trials being given to Canadian magnesite. The increase in the shipment of both calcined and crude magnesite during 1915-16, bears testimony to the usefulness of this material for steel furnace linings. Mr. F. R. McKune, of the Steel Company of Canada, has conducted several practical tests and has introduced a method of mixing pulverized blast furnace slag with Canadian magnesite, and so introducing the necessary iron binder. It is found that the lime present gives no trouble, at least in open hearth steel furnaces. A similar mixture is used with Californian magnesite using only half as much slag. In other cases the dead burned magnesite is mixed with some calcined material and this again with a little sulphate or chloride of magnesium. The plastic mass is applied to the walls, etc., of the furnace and allowed to set 24 hours. When it has set a very gentle heat is applied for 24 hours and finally the furnace is brought to its maximum temperature.

A small amount of refractory brick is made from Canadian magnesite in combination with other materials. The greater part of the refractory use of Canadian magnesite is still, however, in the form of granular magnesite, the result of blending the Canadian article with a suitable iron content and other magnesite lower in lime, and dead burning. One of the most interesting products from Canadian magnesite is metallic magnesium made by the electrolysis of the chloride. Both the solid and powdered magnesium are produced.

Reference has been made to the hydro-magnesite of British

Columbia. The following is an analysis made by the writer on a sample received recently:

	Analysis of Crude	Analysis of Calcined
Silica	2.10%	4.81%
Ferric oxide	.15%	.32%
Alumina	.30%	.65%
Lime	.05%	.11%
Magnesia	41.00%	94.11%
Loss in ignition (CO_2H_2O and organic)	56.4%	———

The calcined product is of a fine white colour, not creamy as is the case with the Quebec magnesite. A portion of the calcined material was taken and a sample of magnesite flooring made from it, using Canadian magnesium chloride produced from Quebec magnesite. The flooring was thoroughly satisfactory both in appearance and strength.

The deposit in British Columbia is the result of evaporation from magnesium bearing waters, and occurs as a fine powder varying from a few inches to six feet in depth, while one deposit is credited with being over thirty feet deep. The hydro-magnesite covers many acres, and is situated at Atlin. There is a direct waterway to the coast, and some hundred tons of this material has been shipped to Vancouver and part of it finally to England, where it has been made into satisfactory magnesite products. Active steps are now being taken with a view to placing this trade on a profitable basis.

Mention has been made of chloride of magnesium as being used together with calcined magnesite for flooring. This naturally suggests the manufacture of the chloride from Canadian rock. Experiments were conducted on a manufacturing scale with the result that the chloride of lime produced by the treatment of the calcined magnesite with hydrochloric acid was found to be no detriment. Research is being conducted with a view to producing chloride of magnesium without the use of hydrochloric acid, as under present conditions the price of hydrochloric acid makes its use prohibitive for this purpose.

Another article that has been made on a manufacturing scale from Canadian magnesite is sulphate of magnesium or Epsom salts. It has been practicable to separate the lime by settling the acid liquor obtained from treating the calcined rock with sulphuric acid so completely as to leave only a trace of lime in the final product.

As regards the trade future for Canadian magnesite, when conditions shall again become normal, the facts must not be lost sight of that in comparison with the Austrian and Grecian magnesites, the relatively high lime content of the Quebec product is a serious detriment, while also iron must be introduced into the Canadian magnesite, whereas with Austrian magnesite this is not necessary. In bidding for trade, foreign competitors will doubtless make the most of these facts to discredit the Canadian product. It must also not be forgotten that the Austrian deposits are particularly well developed and are equipped with the very latest machinery, and in that country labour is cheap. Again in respect of the refractory trade, Californian competition is to be reckoned with; for, as already mentioned, the nature of the magnesite found there is strikingly similar to that of the Austrian product—a feature that will commend itself to consumers. Moreover a great deal of capital, it should be noted, has been invested in the development of the California deposits during the past two years; and under peace conditions, and providing shipments can be made dependably via the Panama Canal, California magnesite could be delivered in the eastern American markets to sell at a price low enough to compete with the products of Austria or of Canada. Although as high a price as from $20 to $25 per ton was realized for Austrian dead burned magnesite towards the end of the time when it was obtainable, the normal price is about $16 per ton. This price ruled when there was no serious competition; and hence it is reasonable to suppose that this price would be reduced if necessary to defeat competition. It remains, therefore, for Canadian producers to reduce their operating costs to a minimum by the use of improved machinery and the provision of adequate haulage facilities, so that they will be in a position to sell their product at a price sufficiently below that asked by foreign producers to outweigh the superior advantages possessed by the

latter having regard to the requirements of the refractory and composition flooring trades.

DISCUSSION

MR. H. J. ROSS, Vice-President of the North American Magnesite Company, delivered a short address descriptive of the operations of this Company at Calumet, Que., and expressed confidence that this industry would continue to be conducted profitably in Canada after the war.

MR. HOWELLS FRECHETTE:—In this connection a brief account of the work on magnesite on which I have been engaged for the Mines Branch, at Ottawa, may be of interest to members. Last year I read a paper before the Institute on "Canadian Magnesite." In discussing that paper Dr. Stansfield suggested the possibility of calcining the magnesite at such a temperature as to drive off the CO_2 from the magnesium carbonate and leave the calcium carbonate unchanged. In this manner, the harmful properties of the lime would be overcome when the calcined magnesite was used for certain purposes. With this in view, I began experiments on samples of magnesite containing about $7\frac{1}{4}\%$ of lime, to determine the range of temperature at which this differential calcining must be carried out. In the course of this work it was conclusively demonstrated that the lime of the magnesite occurred entirely in chemical union with part of the MgO in the form of dolomite. That is, the crude magnesite was a mixture of the two minerals, magnesite and dolomite. The magnesite begins to give off its CO_2 at a little above 500°C., and the dolomite at about 700°C. Thus, the maximum range of temperature is 200°C., but commercial calcining should be done between temperatures of 650°-700°, too narrow a margin for industrial kilns. Had the lime occured as calcite, dissociation of the carbonate of lime would take place at or above 810°C., allowing a much wider latitude for furnace temperature. After calcining the white magnesite, it was noticed that the bulk of it had changed colour—in some cases to a pink, in some to a green, depending probably upon the nature of the furnace atmosphere, and the temperature attained. Scattered throughout were grains of white. I separated these by

careful sorting with the point of a knife, and had the white and the pink grains analysed. The pink were found to be practically pure magnesia, and the white, calcined dolomite. On slaking the calcined material with water, the pink portion broke apart into grains, which were hard and gritty, while the white slaked into a smooth paste that broke up into a milk when agitated with water. By draining off the milk, the bulk of the dolomite was removed, leaving the magnesia in a fairly pure state. The dolomitic lime could be saved by allowing it to settle, and draining it. Using a calcined magnesite containing about 16½% CaO, by small laboratory apparatus, I reduced its lime content to 5.4%, and obtained a product from the washing containing 44.8% CaO and 49.8% MgO. (True dolomite lime gives 58.3% CaO and 41.7% MgO.) With a proper selection of machines, I do not hesitate to say that these results can be improved upon greatly.

MR. J. R. DONALD:—The peculiar feature in connection with the Quebec magnesite deposits is the fact that the four quarries at present being operated are all situated within a mile or two of each other and are practically all in a straight line. The lime runs anywhere from 8% upwards. Good shipping material should not contain more than 10% of lime. It is very difficult to judge by the eye the quality of the magnesite. In general it is white in colour, crystalline, and fine in grain. There are many exceptions to this, and in some cases the dark material, which is very much the colour of a weathered limestone, is quite the equal in quality of the white. I have had occasion to examine a great many dolomite deposits, both along the line of the C.P.R. near Mt. Laurier, Announciation, etc., and along the line of the C.N.R. near Arundel. All of these other deposits have been of dolomite—although some of them showed as high as 30% of magnesia. The material is being shipped very largely to steel works and to the Harbeson & Walker refractories. It is claimed by manufacturers of materials that a magnesite brick cannot be made from rock containing over 8% of lime.

I had occasion some time ago to make an investigation along the lines referred to by Mr. Frechette. My experience corroborates his figures, it being found that after calcining at a temperature below 700°C., that the loss on ignition which

remained was about double that which would be required for the lime content to figure as calcium carbonate. This therefore pointed to the fact that the lime present in the magnesite existed as a dolomite. I might add, it has been found that there is a second point where the magnesia content of the dolomite breaks down giving off its CO_2 and the lime still remains as limestone. This has been proven on a practical scale when magnesite had been calcined for the flooring trade which requires that no free lime shall be present.

MR. ROAST:—In reply to Mr. Frechette's statement that the lime of the magnesite occurred *entirely* in chemical union with part of the magnesia, and that it was not commercially feasible to decompose the magnesite without decomposing the lime, I may state that this does not agree with my experience. The acid test if carefully applied to many samples of magnesite will show local areas which effervesce, indicating the presence of free calcite. Several years ago I had occasion to conduct a series of calcining tests on a commercial scale, and found very little difficulty in liberating practically all the carbonic acid gas from the magnesite and at the same time retaining all the lime in the untouched form of carbonate of lime. As a result of these operations hundreds of tons of Canadian magnesite were so calcined and the resultant calcined magnesite successfully used in the composition flooring trade. I have also made tests on a commercial scale along the lines referred to by Mr. Frechette as to calcining and slaking. While I agree with him that by such means the lime content can greatly be reduced, yet I do not believe the process can be made commercially profitable.

MR. M. E. WILSON:—In the summer of 1900, Reverend W. P. Boshart in the course of a visit to Mr. Donald McPhee, lot 15, range 9, Grenville township, observed a boulder lying near Mr. McPhee's house, which had a whiter and more glistening appearance than the ordinary crystalline limestone of the district, and sent a specimen of the material to Mr. W. B. McAllister of Ottawa, who took the sample to the Geological Survey where it was determined to be magnesite. Learning that magnesite was of commercial value a search for the mineral in place was undertaken by Messrs. McAllister and Boshart with the result that

outcrops and boulders of similar material were found in numerous localities in the district. Later in the season M. R. L. Broadbent of the Geological Survey visited these localities and descriptions of the deposits and the results of analysis of numerous samples collected by Mr. Broadbent were published in the Annual Report of the Geological Survey for that year. The Geological Survey, thus, played an important part in the discovery of the Grenville magnesite deposits.

During the field season of 1916, the magnesite deposits of the Grenville district were again examined by the Geological Survey, and some of the most important results of this investigation may be of interest in connection with the papers of Mr. Roast and Mr. Ross.

The magnesite deposits all occur in extensive outcrops or groups of outcrops protruding through the marine clay and sand, which in the Grenville district as elsewhere in the Laurentian highlands adjoining the lower Ottawa and the lower St. Lawrence, forms flats in the bottoms of the major valleys. They are mainly composed of magnesite, dolomite and serpentine, but also include small quantities of diopside, talc, phylogopite, quartz and other minerals.

The magnesite contained in the deposits is a glistening cream white to milk white or grey material occurring in extensive masses associated with bands or lenses of dark green to light yellow serpentine. Serpentine also occurs disseminated in the magnesite in places, and the magnesite nearly everywhere contains more or less included dolomite; and, since dolomite ($CaCO_3$ $MgCO_3$) contains 30 per cent of lime the magnesite generally contains a certain amount of this constituent, the percentage present varying in proportion to the amount of dolomite which the magnesite contains. In a few localities the dolomite included in the magnesite is more coarsely crystallized and is whiter in colour than the surrounding material, and can be distinguished in this way; but, throughout the great mass of the deposits, the magnesite and dolomite are so similar in appearance that the presence of the dolomite is difficult to detect.

The intimate manner in which the dolomite is disseminated through the magnesite is made evident in several ways. Where the magnesite outcrop had been exposed to atmospheric agencies,

the dolomite being more soluble than the magnesite dissolves away more readily, so that the weathered surface presents an irregular pitted appearance, the magnesite forming the prominences and the dolomite occupying the bottoms of the depressions. The presence of the dolomite in the magnesite can also be detected by treating the mixture with cold concentrated hydrochloric acid, effervescence occurring when the dolomite is present. The relationship of the dolomite to the magnesite can best be observed, however, in the material after it has been calcined in a kiln or furnace without access of air (deoxidizing atmosphere), the dolomite assuming a white and the magnesite a pink colour as a result of this operation. In these specimens the dolomitic portions appear to be included in the magnesite and possess a most irregular outline.

The proportion of magnesite free from dolomite, or dolomite free from magnesite in the deposits, is exceedingly small, the larger part consisting of an intimate mixture of magnesite and dolomite in varying proportions. There are considerable masses of material however, in which the lime content resulting from the presence of the dolomite ranges from 3% to 12%. It is the practice at the mines at present to mine only these fractions of the deposits, so that a product is obtained averaging from 6% to 10% lime.

Up to the present time, deposits of magnesite have been found in four principal localities in the Grenville district. At all of these points there is present extensive masses of marketable magnesite in which the proportion of serpentine and highly dolomitic magnesite, to be removed by cobbing or on mining the material, is less than 40 %. It is estimated from an examination of the surface outcrops of the deposits and from the results of diamond drilling operations that nearly eight hundred thousand tons of magnesite, averaging from 7% to 10% lime, is known to be present in the district.

MR. JOHN A. DRESSER:—As to the question of general production it seems to me the most important phase of the topic immediately before us is whether these deposits are limited to the single area already worked or may they be looked for far afield. The general formation in which these deposits occur is very widely spread. Much prospecting seems to have been done

apparently without results outside of the particular area already described. I would like to enquire of Mr. Wilson whether the geologists, who have worked recently in the district, have found any definite route of determining from the origin of this deposit what the possibilities are of an extension of or of similar bodies elsewhere. Grenville limestone is widespread and it does commonly not look essentially different from the rock in this particular vicinity. Are there any structural features that may give a clue to the size of the present deposits? Perhaps Mr. Wilson can tell us what has been done to determine the origin and extent of these deposits.

Mr. M. E. Wilson:—Complete answers to the questions raised by Mr. Dresser would involve nearly the whole of my report on the magnesite deposits. That there are undoubtedly deposits of magnesite in the Grenville district which have not been discovered seems evident from the facts, that, the direction of glacial movement in the region was approximately from north to south so that boulders of magnesite found in the district must lie approximately to the south of the deposits from which they were derived, and that boulders of magnesite were observed in the district which lie several miles to the east of the known deposits and therefore could not possibly have been derived from any deposit of magnesite hitherto discovered in the region. In districts outside the Grenville area deposits of coarse white dolomite similar to that associated with the Grenville magnesite deposits are known to be present, and this indicates that the process by which the magnesite deposits were formed were active throughout a much wider extent of territory than that in which the magnesite has been found. It is probable, therefore, now that the character and value of magnesite has become known, that magnesite may be discovered in other localities throughout that section of the Ottawa valley in which the rocks of the Grenville series are found.

The rocks of the Grenville series with which the magnesite deposits are associated have all been so greatly deformed that the limestone members of the series, which were originally continuous beds, now occur in crumpled and detached masses and it is only in exceptionally favourable localities that the structure of

this complex can be worked out. Since it is believed that the magnesite deposits have been formed by the replacement of the limestone member of the Grenville series, the structure of these ancient sediments has a bearing on the extent of the magnesite deposits only in so far as it can aid in the determination of the extent of the limestone masses which the magnesite deposits replaced.

The principal structural features exhibited by the magnesite deposits themselves are parallel planes of parting resembling planes of bedding, parallel banding in many places in the deposits, a lenticular structure, faults, slickensided faces, crumplings, and other evidence of deformation. The banded structure exhibited by the deposits arises in part from variations in the colour of the magnesite and in part from variations in the proportion of disseminated serpentine which it contains. The lenticular form commonly observed in the deposits is probably the result of deformation, certain portions of the deposits of slightly different composition or physical character being more resistent to deformation than other portions of the deposits.

As regards the origin of the magnesite deposits, it has been concluded from a study of their character and relationships, that they have been formed by the replacement of the crystalline limestone member of the Grenville series through the agency of solutions emanating from a series of pyroxenic igneous intrusives, known as the Buckingham series. The principal evidence on which this conclusion is based is (1) that the crystalline limestone member of the Grenville series commonly contains disseminated grains of serpentine and serpentinized diopside arranged in bands similar in every respect to the banding exhibited by the magnesite deposits; (2) that the Grenville limestone commonly contains masses of serpentine or diopside partially altered to serpentine which are very similar to those found associated with the magnesite deposits; (3) in many localities throughout the region in which the Grenville series is found, there are also extensive masses of rock, generally known as 'pyroxenite', composed of diopside, scapolite, phlogopite, apatite, feldspar, quartz, calcite, tremolite, actinolite, serpentine, vesuvianite, and other minerals of similar character. It is believed from the association of these masses with limestone, the abundance of lime silicate and

pegmatitic minerals which they contain and the high lime content of the deposits as a whole that they have been formed from the limestone through the agency of pegmatitic magnesia-rich solutions, and the association of diopside and phlogopite with the Grenville magnesite deposits indicates that there is a genetic relationship between the magnesite deposits, the masses of diopside and serpentine found in the limestone of the Grenville series, and the metamorphic 'pyroxenite', and that they have all been formed in some way by the transformation or replacement of limestone; (4) in places masses of dolomite occur associated with the serpentine contained in the limestone sand in a few localities, the dolomite can be seen to be distributed irregularly along fractures traversing the limestone; and (5) the rocks of the Grenville series have been intruded almost everywhere by bosses and thin *lit par lit* injections consisting of pyroxene gabbro, pyroxene diorite, pyroxene syenite, diorite pegmatite, and syenite-pegmatite, (Buckingham series), and it would, therefore, seem probable that the solutions by which the limestone was transformed into diopside and other minerals emanated from these intrusives.

THE PARTRIDGE SMELTING FURNACE.

ITS CONSTRUCTION AND ADAPTABILITY.

By Allen R. Partridge

The new Partridge Smelting Furnace is a mechanical device intended for smelting ores on a small as well as on a large scale. Its success in small units makes it an invaluable machine for small and remote properties and especially in new countries, sparsely populated and lacking in transportation facilities. The smaller units with a capacity of five to thirty tons in twenty-

The Partridge furnace in operation.

four hours, can be transported on pack animals and they are just as effective as a large furnace.

The construction of the furnace is quite distinct from the present standard type, although the jacket, where the actual smelting takes place, is similar to that of any other furnace except as regards the method of introducing the air blast. The furnace consists of a double steel cylinder resting on a sheet steel bottom plate, and having a double top cover of steel (see Fig. 2, A). The jacket, which is water cooled, rests on a frame of I beams set between the two cover plates and is open to the interior of the cylinder. (Fig. 2, B.) Two large doors diametrically opposite to each other are placed on the cylinder. (Fig. 2, C.) Inside the cylinder a large conical plate of cast iron revolves slowly upon suitable ball bearings. (Fig. 2, D.) Above this plate is another frame which imparts rigidity to the whole structure and carries a ball-bearing jack-screw. (Fig. 2, E.) This is made fast to a cast-iron platform upon which the crucible, or inner hearth of the furnace, is set and raised into the operating position. When this is done the front door is put on and the crucible made fast to the door by means of bolts. (Fig. 2, F.) A matte spout is then slipped into place so that the metal can be drawn from the crusible ih the ordinary manner. The method of starting the furnace is unusual, yet very simple:—The cold, but perfectly dry brick-lined crucible is run on the cast-iron platform, previously mentioned, and raised into position, so that the top of the crusible shall be within two inches of the bottom of the water jacket. The door is then put on, the crucible made fast to the door and the matte spout put in place. At this stage water is turned oe in the jacket.

The man on the feed floor now drops a little oil-soaked waste, which he has previously ignited, through the feed door and into the crucible, and on top of this he drops a few sticks of wood-kindling taking care not to smother the fire. When the kindling is burning freely, more is added until it is some six inches above the opening between the crucible and the jacket which will be referred to hereafter as the 'tuyere.' The kindling is allowed to burn freely, and when well ignited, a bed of coke or charcoal 18″ deep is fed into the furnace. The foreman now watches the tuyere through mica peep holes set in the

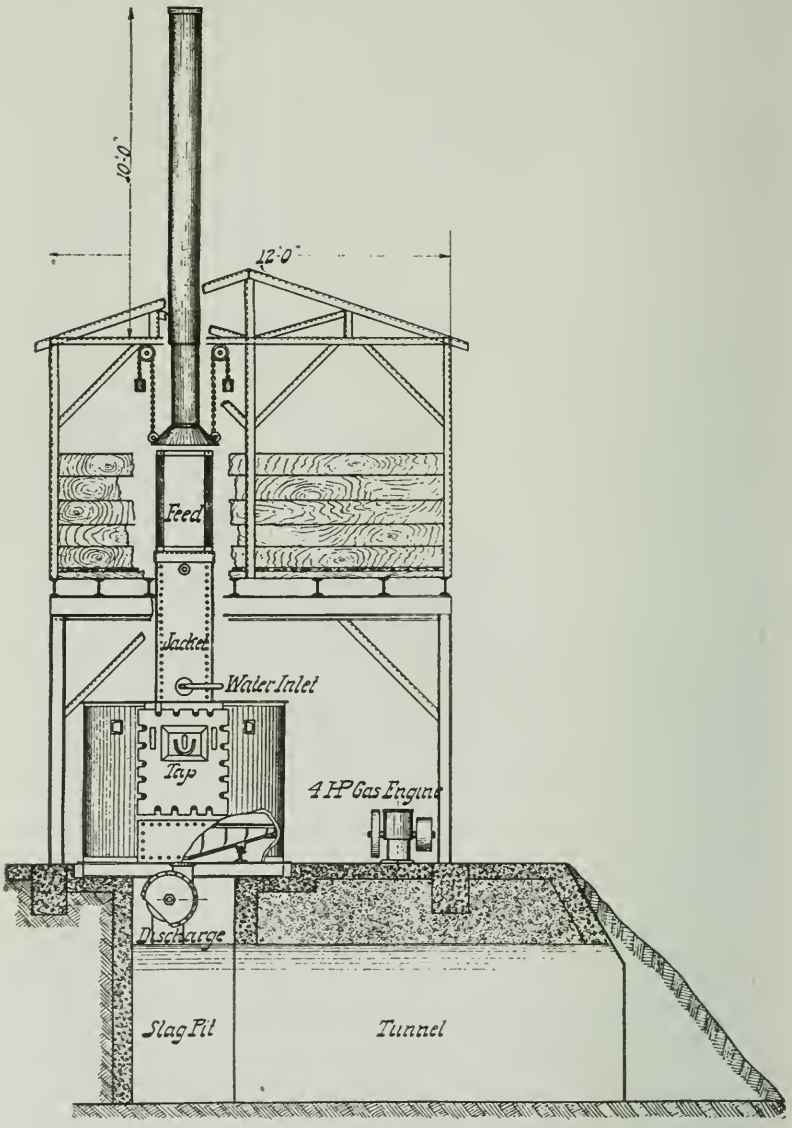


Fig. 1

Partridge Hot Blast Smelting Furnace.—Tuyere Area 11" x 18"
Capacity: 12 tons of charge in 24 hours

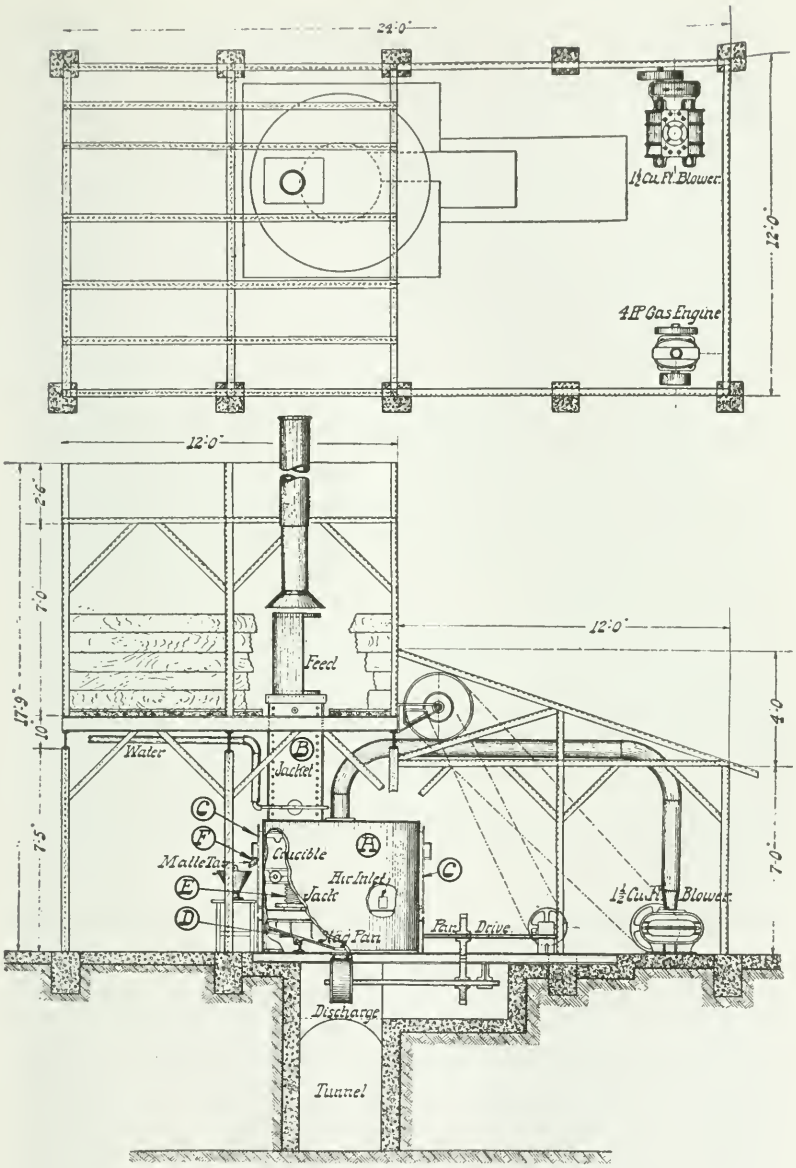

24'0"
1½ Cu. Ft. Blower.
12'0"
4 H.P. Gas Engine
12'0"
12'0"
2'6"
7'0"
17'9"
10'0"
7'5"
Feed
Water
B
Jacket
C
F
Malleable
E
D
A
Crucible
Air Inlet
Jack
Hog Fan
C
12'0"
4'0"
7'0"
1½ Cu. Ft. Blower
Fan Drive
Discharge
Tunnel

Fig. 2

cylinder, and when the kindling is consumed he will find that coke has replaced kindling and is burning brightly. He then orders the charge man to feed one charge of barren slags, and turns on the air blast.

The air from the blower passes from the bustle pipe over the top of the cylinder between the two covers, then between the double sides of the cylinder into the cylinder itself through four openings. From the interior of the cylinder it enters the jacket by the tuyere previously described, causing a fierce fire. In less than two minutes after the blast has been turned on, the slag charge will be seen to pass the tuyere on its way to the bottom of the crucible. As soon as the slag begins to run out of the tap hole, which has purposely been left open, the foreman tamps the tap and orders the charge man to proceed with the regular charges of coke (of charcoal) and ore charge until the jacket is full to the top. The foreman now draws off a couple of pots of slag from the crucible to heat it well, and the furnace is in full operation. This process of blowing in takes from thirty to forty-five minutes.

The moveable curcible, two of which are supplied with each plant, permits the furnace to be operated intermittently, and in the event of a freeze, the defect can be corrected in less than an hour by withdrawing the crucible, bearing down the frozen charge, and substituting the spare crucible. The main trouble is in deeing that the foreman shall make this change as soon as he finds that the furnace is not running properly and does not try to doctor the charge; as this means loss of time and money. The 'blowing out' of the furnace is accomplished by running the charge down entirely, and it is well to add as a final wash a charge of barren slags.

While the smelting is proceeding, the slag flows from the crucible by a spout, and drops on the revolving cast-iron plate. The air-blast from the blower is forced on this plate and readily absorbs the heat evolved from the very thin layer of slag travelling on the rotating plate. Stationary scrapers set at an angle push the slag to the centre of the plate where it is finally extracted automatically from the furnace through a revolving air trap discharge. It will readily be seen that the blast becomes heated by this means and will enter the jacket at a high

degree of temperature usually 380° to 450° C., depending on the composition of the ore and the charge.

These mechanical devices were studied and perfected with the idea of simplifying the manipulation of the blast-furnace in general, especially in its application to small units, Their use has solved the small furnace problem remarkably well and has also developed several new features in the art of ore smelting. One of the principal reasons for the notable success of the plant is the method of introducing the blast into the zone of fusion. Small water-jacket furnaces have given great trouble in operating due to the forcing of the blast into the jacket by means of tuyeres of comparative small area and at a high pressure in order to introduce a sufficient volume of oxygen to ensure proper combustion and the blast has invariably been introduced at atmospheric temperature. In the case of the Partridge furnace, the tuyere area is fully five times greater than in standard practice, hence it is possible to smelt with a larger volume of air at an incredibly low gauge pressure. The aim is to introduce the air with sufficient velocity to ensure a high degree of combustion and at the same time to avoid the usual heavy commotion inside of the jackets. In this furnace the gauge glass will show pressures ranging from four ounces to the square inch, up to twelve; the variations being due to the density of the charge, the height of the column, and to some extent to the size of the stack.

Together with the low blast-pressure we have the crucible with its internal overflow of slags. This crucible makes as perfect a settler as has been devised so far. The molten mass of charge trickles by the tuyere into the crucible which is kept very hot both from the superheated blast and slag radiation. This keeps the molten material in a remarkably fluid state and permits the metals to separate from the slags to an unusual degree. The slag overflows gently at the slag spout while its surface is almost stationary, and in consequence the slags are remarkably free from metallic shot. The use of a large volume of air at a low pressure confines the zone of fusion to a very small depth, right above the tuyere, and ensures most marked and perfect chemical reactions and the absence of top firing. The use of the settling device surrounded by a great heat enables

smelting to be carried out in very small jackets with the additional advantage of being able to produce slags of very varied composition. In copper smelting, slags of the following compositions will run freely and the highest recoveries can be made.

FeO	16% to 55%
SiO_2	27 %to 62%
CaO	0 to 35%
Zn	0 to 20%

In lead smelting more care has to be taken in the charge composition and very high silica contents entails some lead losses. A study of the above range of slag composition will show that the furnace is the best adapted for individual mines where close slag regulation is out of the question. The recoveries in precious and semi-precious metals are very high, and volatilization very low, in fact, usually negligible. Copper slags seldom carry 0.01% of copper and no gold or silver, while lead slags rarely contain over 0.4% of lead with no gold or silver.

The large volume of oxygen which can be introduced at will permits the furnace to be substituted for the Bessemer converter and used as such. Again the large volume of preheated blast at a low pressure makes this furnace an ideal pyritic plant. An ore containing:—

Cu	1.3%
S	38%
FeO	42%
Zn	4%
SiO_2	5%
CaO	3%

with the addition of 15% of silica and 8% of combined lime was run pyritically with perfect success. The furnace was started on a coke charge 6% which was cut to 3% on the second charge, 1½% on the third, 1% on the fourth, ½% on the fifth charge and nothing on all subsequent charges. The furnace ran better and hotter on the no fuel charges and produced a matte assaying 17% in copper on the first run. This was

accomplished at Stege, California, in a furnace having an area of 11″ x 18″ and a capacity of 1,200 lb. per hour.

The Partridge furnace is the invention of the writer who conducted his experiments in a remote section of Mexico, where an apparatus of this kind was necessary for success. The writer was in charge of a group of mines carrying unusually rebellious ores in south-western Tepic and the topography of the country made sectional machinery imperative. A general analysis of the ores was as follows:—

SiO_2	34%
Fe	20%
S	22%
Zn	6%
As	8%
Sb	2%
Pb	1%
Cu	0.25%
CaO	1%
	94.25%

Silver.....28 oz. per ton
Gold..... 0.9 " " "

Hyposulphate leaching, after chloridizing the ores in hand-reverberatories, proved successful but offered many objections such as high cost of treatment, difficulty in keeping down the volatilization losses during the chlorination roast, and the necessity of a second treatment to recover the gold.

This ore was smelted in the Partridge furnace with the addition of 10% of lime rock ($Ca\ CO_3$) and 2% of carbonaceous fuel. Coke as well as charcoal was used and as charcoal was by far the less expensive fuel, it was adopted. Two runs were made prior to shipment of the bullion. The first run was on raw ore and yielded an iron matte carrying about 250 ounces per ton of silver and 8 ounces of gold or a concentration of 9 tons of ore into one of matte. This matte was subsequently resmelted with selected silicious ores and further concentrated to about 10 into one or 2,500 oz. of silver and 80 oz. of gold

per ton, with some 20% of copper. In both smelting operations the slags were astonishingly clean, carrying less than 0.02 ozs. of silver and no gold. The cost of operation was $1.25 per ton for the raw ore smelting and $2.00 per ton for the rematting smelt.

In contrast to the above citation, the writer installed a plant at Magdalena, State of Oaxaca. The ores from these mines are self-fluxing lead carbonates with some galena containing 22% of lead and 35 to 60 ounces of silver. The lead and silver are recovered as a 98% bullion, which is cupelled on the spot; the silver being sent to the mint, and the litharge sold locally to pottery makers. The usual by-product of speiss carries about 12% of lead and 14 oz. of silver. It is heap-roasted and used as iron flux for subsequent operations. The slags are rather basic and carry 0.4% of lead with a trace of silver, 10% of charcoal is used with the charge. Both the Tepic and the Magdalena furnaces have a daily capacity of 10 metric tons.

THE OCCURRENCE AND TESTING OF FOUNDRY MOULDING SANDS[1]

By L. Heber Cole

Annual Meeting, Montreal, March, 1917

The need in Canada for foundry moulding sands of different grades suitable for different classes of castings has increased greatly in the last few years, and has led the Mines Branch to investigate many Canadian sand deposits to determine their suitability for this class of work. At the present time a large part of the sand used in Canadian foundries is imported; and, although, in a number of places, local deposits furnish small quantities to foundries in the immediate neighborhood, no deposits have been opened on such a scale as to furnish properly graded sand to the foundry trade of Canada; and the supply for this market is now being drawn mostly from the United States.

In the summer of 1914 the investigation of the sand deposits of Quebec by the Mines Branch was commenced, the field work being continued during the seasons of 1915 and 1916. The scope of the investigation has since been extended into eastern Ontario. It is hoped that, during the coming season (1917), field work will be carried on in western and southwestern Ontario.

In the course of the regular field work so far accomplished, several deposits of sand were encountered, which, based on the field examination, gave promise of being suitable for moulding sands. Samples of these were taken and sent to the Mines Branch Laboratories, Ottawa, for examination and testing.

Foundry Sands in General.

Foundry sand may be divided into two main classes:

(1) Moulding sands, or the sand which is actually used to make the mould, into which the molten metal is poured; and,

[1]By permission of the Director Mines Branch, Dept. of Mines, Ottawa.

(2) Core sands, which are utilized for making the cores that occupy the hollow spaces in the casting.

The material used for foundry sands varies greatly, according to the nature of the casting, the metal to be poured, the part of the mould, and the foundry where it is employed. Thus, materials varying from a heavy, clayey loam to a coarse river sand are used according to the nature of the casting being made. A sand which is suitable for a coarse casting would not be satisfactory for fine work; frequently also an entirely different grade of sand is used for making the cores. Again, in green sand moulding the sand used differs from that used in dry sand moulding, where the moulds and cores are first baked. The practice in various foundries is so diverse and the sand employed for different grades of castings varies so widely that it can readily be seen that it is almost impossible to lay down a hard and fast set of standards to which a sand must conform, in order to be called a foundry sand. This is partly due to the manner in which the average foundry man looks on his sand, and partly to the lack of definite knowledge as to the behaviour and action of certain sands with relation to the castings made in them. The sand used in most cases is employed on the advice of the head moulder in the foundry, who generally trusts to his experience in handling sands as to the suitability of a particular sand for the work in hand. The appearance of the sand to the eye, and whether it will retain the impress of the hand when damp are generally all the tests to which the sand is subjected. If the moulder does not like the look of the sample submitted, and, especially, if he knows it to be a local sand, it is frequently condemned as not suitable without further examination. While this condition of affairs exists, it will be hard to adopt any standards for moulding sands, but it is conceivable that, when the qualities of different sands in relation to the class of castings made in them have been more fully studied, it will most likely be possible to formulate by a series of laboratory and foundry tests, a set of standards with limits within which a sand may be determined to be suitable for a certain class of casting by systematic laboratory and foundry tests.

The Occurrence of Moulding Sand.

1. *Natural Moulding Sands.*—Moulding sands occur in two main types of deposits, but variations of these types may be encountered. These are: (a) from flood plain deposits; and (b) re-washed ancient beach sands.

In respect of the first mentioned deposits: from the nature of a moulding sand, it being essentially a silica sand with each individual grain coated with a bonding material, one would expect to find it occurring where deposits of sand and clay were constantly being intermingled and worked over by water, and, therefore, moulding sands in flood plain deposits are of quite common occurrence. In these beds the sand and clay have been well and intimately mixed by the river currents, and deposited on the higher levels in flood time, the excess of clay, being more minute, is carried off by the water. One would thus expect to encounter moulding sands of this character along the upper terraces of the larger rivers of the country, such as the St. Lawrence; also along the banks of the ancient waterways.

The second class of deposits which are frequently encountered are of secondary origin. The sand bars and beaches of the ancient seas have been worked over by the waves as they recede, when new levels of the lakes and seas are formed. The washed material from these beaches consists of sand and clay, the former being deposited in greater abundance. It is in deposits of this class, which are found at a lower level than the old beaches or water margins, that moulding sand may be expected to occur. These deposits are, therefore, to be looked for in the vicinity of the ancient glacial lake margins, such as the Iroquois and Algonquin, which formerly occupied the Great Lake Basin; and also within the boundaries of the ancient lake Agassiz in Manitoba. Similar ponded water bodies in glacial times extending as far as the foot hills of the Rocky mountains in Alberta may also have deposits of this character within their margins.

2. *Prepared Moulding Sands.*—The natural moulding sands referred to above comprise by far the greater part of the sand used in Canadian and United States foundry practice; but

there are being employed increasing amounts of what may be termed 'prepared moulding sands', and as the suitability of this class of sand becomes better known there is no doubt that their use will be extended widely. These sands are prepared for use by crushing either sandstones which have a very friable bonding material, or else a decayed granite, or shattered sandstone having the fractures filled with a plastic material (such as kaolin, etc.). These sands have no fillers added to them, but perhaps, have to be screened and washed as well as crushed before being offered to the trade. Under this class may also be placed those earthy loams which, by washing, to remove part of the clayey content, may be utilized as a suitable foundry sand.

3. *Synthetic Moulding Sands.*—Many foundrymen have expressed the opinion that the present known deposits of high grade natural moulding sand will become exhausted, necessitating resort to artificial or synthetic moulding sands made by intimately mixing finely crushed quartz, or clean sharp sand, with clay, so that each grain of quartz would become uniformly coated with the clay. It can readily be seen that sand so prepared will have decided advantages over the sands at present in use, in that it will be possible to manufacture a uniform material for the class of work required, and also that variations in the material can be made at will to meet the requirements of the trade.

THE TESTING OF MOULDING SANDS

The examination and testing of a moulding sand deposit can be divided into two parts: (a) The field examination of the deposit; (b) Laboratory examination and testing of the sand.

Field Examination of a Sand Deposit.—In undertaking a field examination of a moulding sand deposit, there are several points to be taken into consideration:

1. Nature and extent of deposit (area and depth).
2. Uniformity of sand.
3. Transportation facilities.
4. Location, with respect to the larger markets.

The importance of a field examination can readily be grasped. A sand may be suitable for foundry work in every way; but, if it is not in sufficient quantity, easily exploited, and is not favourably situated to the larger markets for this class of material, the deposit is of little value as a commercial venture.

The method of field examination employed by the writer is as follows: The area is tested by drilling test holes with a 6-inch post hole auger drill, in a sufficient number of places to determine the extent and depth to which the sand is encountered. These holes are indicated on a map of the area and the boundaries of the sand plotted. The sand encountered in the drill holes is carefully examined with a hand magnifying glass to note any marked difference in its character. Samples are taken from these holes and mixed together to obtain a uniform sample for testing in the laboratory. If more than one grade is noted, separate samples are taken.

Qualities of a Moulding Sand to be determined.[1]—When commencing the laboratory tests of the samples taken in the field, the question arose, "What are the requirements of a good moulding sand?" The literature on moulding sands so far is very meagre, and any systematic series of laboratory tests to determine a sand's suitability for a foundry sand are confined to work done by several of the State Surveys, the Bureau of Standards, Washington, and the American Foundrymen's Association. The opinions of the investigators appear to differ greatly both as to the series of tests necessary, as well as the method by which they are to be carried out.

From a study of the literature available, and after numerous conversations with practical foundrymen throughout the country, it was found that the qualities to be taken into consideration in the examination of a moulding sand are as follows:

Texture.—The texture or fineness of grain is one of the most important points of a sand. This will necessarily vary, according to the size and kind of casting to be made in it. Hence, it is at once obvious that sands will have to be selected to suit the class

[1] The sands referred to here are moulding sands proper. Core sands vary widely in different localities, and mostly require bonding material to be added to them. They will be treated separately at a later date.

of work for which they are required, or, in other words, sand which is suitable for light work would, perhaps, be a failure when used with heavy work, or *vice versa*.

Refractoriness.—The capability of resisting effectively the destructive action of the heat of the molten metal is of importance. The greater the size of the casting, the longer it will be in cooling, hence the sand in contact with the metal will be subjected to the intense heat for a longer period of time. It is thus obvious that for large castings a more highly refractory sand will be required than for small castings.

Bonding power.—Moulding sands should possess sufficient bonding power or cohesiveness of their particles to each other to retain firmly the shape and form of the pattern, and also to resist the pressure of the molten metal in the mould. This bonding power depends partly on the clay mixed through the sand particles, and the clay coating on the individual grains, and partly on the nature of the grains, whether they are angular or rounded, coarse or fine. As a rule, the finer and more angular the sand grains, the greater the bonding power.

Permeability.—One of the properties of a moulding sand which helps to determine its suitability for foundry use is that of allowing the escape of gases through it. The molten metal develops gases which exert a pressure on the face of the mould and, unless the spaces between the grains are sufficient to enable the gases so generated to escape freely, there will be serious danger of creating scabs or causing the castings to blow on this account. Obviously, then, heavy castings will require a more open sand of a coarser grained texture than will fine castings.

Durability.—The durability or life of a sand is of extreme importance. There are many sands which, when used once or twice, lose some of their desirable qualities, and soon become 'dead' or useless. Obviously, the sand in contact with and adjacent to the molten metal will suffer most. The present practice is to screen out the coarse particles and add fresh sand to the remainder. Hence, the greater the durability of a sand, the better it is, as it will last longer, and it will not be necessary to add fresh sand to it so frequently.

In deciding on methods to employ in examining the samples

taken, in order to ascertain their characteristics regarding the qualities just enumerated, it was decided to proceed as follows:

The sample for examination is first passed through a 10-mesh screen, and what is retained on this screen is considered the sample for examination; and all tests are conducted with this material.

Texture.—The texture is determined by making a granulometric analysis of a representative sample, generally 100 grams. This is passed through a series of Tyler Standard Screens, by shaking thoroughly on a mechanical shaker; the material retained on each screen being collected, weighed, and noted. If the sample in the first place weighs 100 grams, the weight recorded as retained on each screen is the percentage retained on that screen; and the cumulative percentage of all material that would be retained on any given screen, if that screen alone were employed, can readily be determined. The screens used for this test and the form used for tabulating the results are as follows:

Indicate the Screen Crushed through and also First Retaining Screen	SCREEN SCALE RATIO 1.414				Sample Weights	Per Cent	Per Cent Cumulative Weights
	Openings		Mesh	Diam. Wire Inches			
	Inches	Millimetres					
.....	1.050	26.67		.149	.....	.....	.....
.....	.742	18.85		.135	.....	.....	.....
.....	.525	13.33		.105	.....	.....	.....
.....	.371	9.423		.092	.....	.....	.....
.....	.263	6.680	3	.070	.....	.....	.....
.....	.185	4.699	4	.065	.....	.....	.....
.....	.131	3.327	6	.036	.....	.....	.....
.....	.093	2.362	8	.032	.....	.....	.....
.....	.065	1.651	10	.035	.....	.....	.....
.....	.046	1.168	14	.025	.....	.....	.....
.....	.0328	.823	20	.0172	.....	.....	.....
.....	.0232	.589	28	.0125	.....	.....	.....
.....	.0164	.417	35	.0122	.....	.....	.....
.....	.0116	.295	48	.0092	.....	.....	.....
.....	.0082	.208	65	.0072	.....	.....	.....
.....	.0058	.147	100	.0042	.....	.....	.....
.....	.0041	.103	150	.0026	.....	.....	.....
.....	.0029	.074	200	.0021	.....	.....	.....
Pass.....	.0029	.074	200	.0021	.....	.....	.....
					Totals	.....	.....

In order to gain some idea of the relative fineness of the sand, and to be able to express this in one figure, a more convenient form than the whole granulometric analysis, the average fineness of the sample is calculated. The average fineness is determined as follows: The material passing through each screen and retained on the next smaller one is multiplied by the mesh of the screen passed through. The results obtained are totalled and divided by 100, the resultant being the average fineness. In other words, if all the grains of the sample were reduced to a uniform size, they would just pass through a screen whose mesh was equal to the average fineness of the sample. For example. The granulometric analysis of a sand is:—

	Mesh	%	The calculations for average fineness will be:	
Retained on.....	10	.....		
"	14	.....	14 x 0.10 =	1.40
"	20	0.10	20 x 0.12 =	2.40
"	28	0.12	28 x 0.23 =	6.44
"	35	0.23	35 x 0.65 =	22.75
"	48	0.65	48 x 0.72 =	34.56
"	65	0.72	65 x 1.50 =	97.50
"	100	1.50	100 x 13.01 =	1301.00
"	150	13.01	150 x 22.30 =	3345.00
"	200	22.30	200 x 61.37 =	12274.00
Through.....	200	61.37	Total.....	17085.05

$$\frac{17085.05}{100} = 170.85$$

or the average fineness of the sample is 170.85.

Microscopic Examination.—It has been found that the free use of the microscope in examining the sand, both fresh and after being burned, has added valuable data which help materially in determining a sand's suitability for a moulding sand. Microphotographs of the sand are also valuable for comparison.

Refractoriness.—The refractoriness of a sample is determined by preparing a cone of the sand and heating it to fusion in an electric furnace along with standard Segers cones, and noting at what cone the sand fuses.

Bonding Power.—The determination of the bonding power of a moulding sand from tests either in a laboratory or in actual

foundry practice presents many difficulties, and the methods so far devised give only relative results, and these are variable according to the skill of the operator making the tests. The tentative method adopted for these tests is similar, with a few slight modifications, to the method employed by the Bureau of Standards at Washington, D.C. By this test the transverse strength of the sand is determined. A known quantity of a particular sand is weighed (generally 500 grams) and mixed with a definite quantity of water so that the sand will just hold together. From this sand a test bar is made, one inch square, in section, by 12 inches long. This is moulded in a snap flask on a piece of plate glass. The sand is packed in the flask as uniformly as possible from end to end with the thumb and forefinger of each hand, and smoothed with a trowel. The flask is removed, and the test bar is left on the plate. The glass plate and bar are then weighed, the weight of the glass plate having been previously determined, the weight of the test bar is obtained by difference. While the test bar still retains its moisture, it is gently and steadily shoved lengthwise over the edge of the glass plate until it breaks off and the length breaking off is noted. Continuing this operation, successive portions are broken off and the average length of the overhang, at the breaking moment, is determined. The weight of the bar in grams being known, the transverse strength of the specimen can be calculated from the following formula:

$$S = \frac{\text{Wt. of bar (in gms.)}}{4} \times \frac{L^2 \text{ in inches}}{453.6} \text{ where}$$

S = transverse strength.

L = length of overhang in inches.

A series of tests were conducted on each sample, taking 5 cc. additional water each time until the test bar deforms on attempting to shove it over the edge of the glass. An average of the transverse results obtained by using varying quantities of water is taken to represent the transverse strength of the specimen.

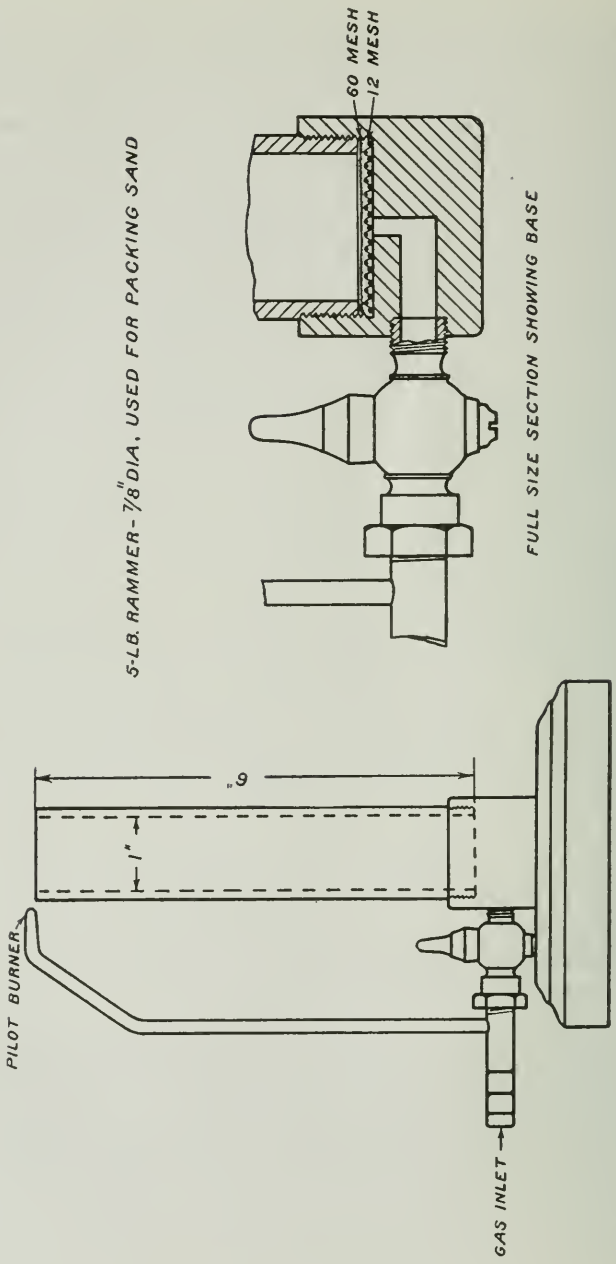


Fig. 1. Permeability Apparatus for determining Permeability of Moulding Sands.

Permeability.—The method adopted for the determination of the permeability of a moulding sand is, as far as the writer knows, entirely new. Investigations along this line have consisted in passing either a definite amount of water or air through the sand, and noting the time required. Both these methods introduced a serious error, since once the first air or water had passed through the sand, channels would be opened through the sand which would greatly facilitate the flow of the remainder of the air or water. To overcome this difficulty the apparatus shown in Fig. 1 was designed, and the results so far obtained have been extremely satisfactory. With this apparatus, ordinary illuminating gas at a definite pressure is passed through the sand, and ignited the instant it reaches the top of the sand tube. The interval of time required from when the gas is turned on to the moment of ignition, is noted by a stop watch, thus determining the initial passing of the gas, and overcoming the error due to the forming of channels through the sand. Three tests are run on each sample, using fresh sand each time. The average time is taken as the permeability factor. The sand is packed uniformly in the cylinder by placing in a small quantity at a time (about 1 inch of sand), and pressing it down for 5 seconds, using a 5 lb. weight. When the cylinder is filled, it is struck off flush on the top with a straight edge.

Durability.—All the foregoing tests may be regarded as being in the nature of a preliminary enquiry to determine whether the sand so tested can be classed as a moulding sand and passed on to the durability test, which must always be regarded as the final one by which a sand from an unworked deposit is accepted or rejected. The durability test is made under actual working conditions, hence definite knowledge of the usefulness of a sand can be determined.

If a sample when subjected to the previously described tests, has proven satisfactory, within certain accepted limits, a larger sample is obtained, and tested in a commercial foundry, where castings are being made of the class to which the sand is deemed best suited. A pattern is chosen, generally one which comes in the general run of the foundry, and the sand to be tested is used with this pattern. Care is taken to keep separately the sand in which the casting is made, and another

cast from the same pattern is made, using this sand without adding any fresh sand to it. This operation is repeated until the sand shows signs of becoming 'dead.' After each cast, the sand is thoroughly mixed and a sample taken. These samples are submitted to all the laboratory tests previously described, and any differences are noted. For means of comparison, a duplicate series of castings is made in parallel with the sand being tested, employing some well-known moulding sand.

RESULTS OF A TEST ON A MOULDING SAND COLLECTED NEAR BROCKVILLE, ONT.

To illustrate the methods employed in the examination and testing of a moulding sand the following report of a test on a deposit of moulding sand from near Brockville, Ont., will serve. This deposit was one encountered in the course of the regular field work, in connection with the investigation of the sand deposits of Ontario.

Field Examination.—The deposit in question lies $2\frac{1}{2}$ miles to the west of the town of Brockville, Ont., between the G.T.Ry. line (Montreal to Toronto) and the river road, (Brockville to Belleville.)

As far as could be determined in the time at the disposal for the field examination the area underlain by moulding sand is of considerable extent, although detailed work was only carried out on the area shown in the sketch map. Fig. 2. No time was available to trace the extension of the deposit to the eastward or to the west, but this will be done during the field season of 1917.

The topography of the immediate district is decidedly rugged. The drift with which the district is overlain consists of rolling hills of boulder clay, sand, and gravel, through which numerous 'islands' of bare rock protrude. These patches of bare rock consist to the north and northeast of Potsdam (?) sandstone and to the south and west of Laurentian granites. All outcrops of rock examined were well glaciated, and rounded, showing clearly defined striae.

By reference to the sketch map Fig. 2, (p .280) it will be seen that the area so far known to be underlain by moulding sand lies

between and around the rock outcrops already mentioned. A stream passing through the deposit has revealed clay beneath the sand.

The sand lies beneath a thin layer of loam averaging about 6 to 12 ins. thick. In most places where tested there was a definite line of demarcation between the loam and the sand.

The sand is fairly uniform over the whole deposit shown in the sketch, and will average 2 ft. 4 ins. thick. In all the test holes and pits, only two boulders were encountered, each about $2\frac{1}{2}$ ins. diameter, so that the sand appears to be free from stones in the area examined. At the edges of the deposit, where the sand and boulder clay are in contact, it may be that the boulders are more frequent.

In order to determine the nature and extent of the deposit a number of test pits were examined and drill holes bored as indicated on the sketch map. The results obtained from these pits and holes are as follows:—

Results of Test Pits and Borings on Brockville Sand Deposit.

Hole or pit	Amount of stripping	Thickness of moulding sand	Material below moulding sand
1	8 ins.	3 ft. 8 ins.	Sand and clay interbanded.
2	8 "	3 " 4 "	" "
3	14 "	1 " 0 "	Sandy clay.
4	12 "	1 " 2 "	" "
5	10 "	1 " 6 "	" "
6	10 "	3 " 10 "	" "
7	15 "	1 " 9 "	" "
8	8 "	2 " 0 "	" "
9	10 "	1 " 10 "	" "
10	7 "	2 " 2 "	" "
11	10 "	2 " 6 "	" "
11a	Stiff clay to	a depth of 3 ft.	
11b	" "	" "	
12	6 ins.	2 ft. 0 ins.	Sandy clay.
13	10 "	1 " 6 "	" "
14	4 "	2 " 6 "	" "
15	8 "	2 " 6 "	" "
16	6 "	3 " 0 "	" "
17	8 "	2 " 0 "	" "
18 pit	8 "	2 " 6 "	" "
19	10 "	2 " 8 "	" "
20 pit	4 "	3 " 2 "	" "

No idea can be given of the total tonnage of sand available as the complete boundaries were not located, but that there is a considerable quantity there can be no doubt, judging from the area tested.

Preliminary Tests.—A sample of 40 lb. of sand was taken when the deposit was first visited, and this was applied by the Alex. Fleck Limited foundry, Ottawa, to make moulds for three iron castings, one at a time, using the Brockville sand, exclusively, each time. The weight of the casting was about 12 pounds and all three casts were perfectly satisfactory, having a good smooth surface free from scabs, and corners showing clean definition.

This preliminary test having proved satisfactory, two lots of 600 pounds each were dug and shipped by the writer to Ottawa without preparation in any way, in order to test the lasting and wearing qualities of the sand. Care was taken to see that the samples collected were representative of the whole deposit as far as could be ascertained. One shipment was taken to the foundry of Alex. Fleck Limited, Ottawa, where the first tests were made; and the other 600 lb. was delivered at the brass foundry of Lawson Bros., Ottawa.

At both places the tests carried on were made under the supervision of the writer, who followed closely all the results obtained, and examined the sand and castings after each cast.

Test of Sand at the Fleck Foundry.—It was desired to gain an idea of the life of the sand when employed with fairly heavy pieces of casting. In order to obtain comparative results the same amount of fresh No. 3 Albany moulding sand, as used in this foundry, was taken and used side by side with the Brockville sand on the same pattern. After each cast, each sand was kept separate, thoroughly mixed, sampled, and used again. The piece cast was an iron flange in the shape of an L, 5 ft. long, 10 in. wide, and 1 in. thick, with a 3-in. flange 1 in. thick, on one side. The weight of the casting was approximately 200 lb.

Five castings were made in each sand under ordinary working conditions, the moulding being done on the two sands by the same moulder throughout, under the direct supervision of the foreman. Care was taken that each sand was thoroughly mixed

after each casting. A sample of each sand was taken when fresh, and after each burn, and examined in the Mines Branch Laboratories. No fresh sand was added to either test and only the sea coal that was absolutely necessary was employed.

The castings were examined after each cast There was no noticeable difference between those cast in either sand or from the first and fifth cast in the same sand.

Test of sand at Lawson's Brass Foundry.—The sand sent to Lawson Bros. was used in their brass foundry on general run of work, employing whatever pattern they needed each day. The Brockville sand was kept separate throughout and the cast was varied between brass and iron, all castings being small. The sand was mixed thoroughly after each burn and no fresh sand was added to it. Samples after every alternate burn were taken for examination. Seven castings in all were made in the same sand, the weight of castings varying from 12 to 50 pounds. Five were brass and two iron. All castings when examined showed clear, sharp, well defined edges, and the body free from scabs. No sign of burning appeared on any of the castings. A sample of the fresh Albany No. O, as used in this foundry, was taken for comparison.

Laboratory Tests.

The samples for examination—obtained from the casting tests at the two foundries—were subjected to the following laboratory tests:

Granulometric Test.—The samples as brought from the foundries were each treated as separate samples. Each one was thoroughly mixed and quartered, and 100 grams taken, and put through a set of Tyler Standard Screens, on a mechanical shaker. The results obtained are tabulated in Table No. 1.

Transverse Strength.—The test for determining the transverse strength or bonding power of the sand was carried out in a similar manner to that employed in the Bureau of Standards, Washington, with the exception to the procedure followed there of always taking the same amount of water. A series of tests were made on each sample, varying the amount of water 5ccs. each time, and

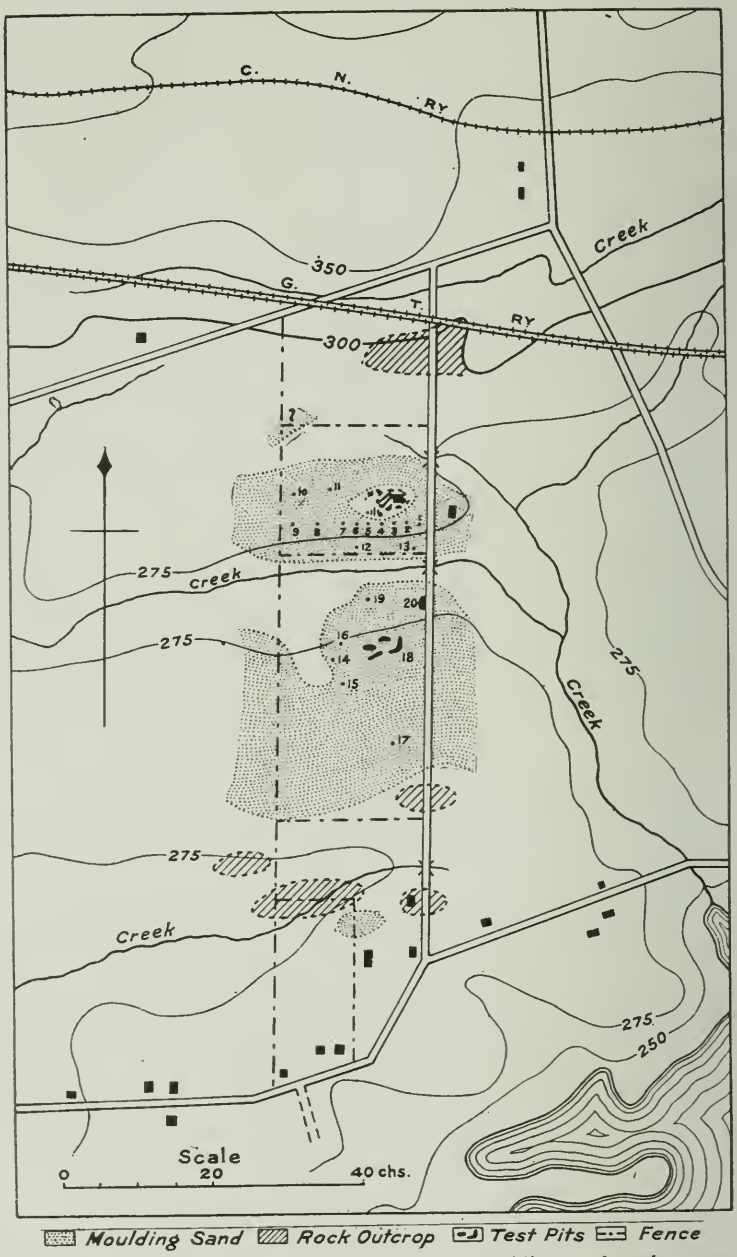


Fig. 2. Sketch map showing areas munderlain by moulding sand on the property of T. H. Bresee, and others, 2½ miles west of Brockville, Ont.

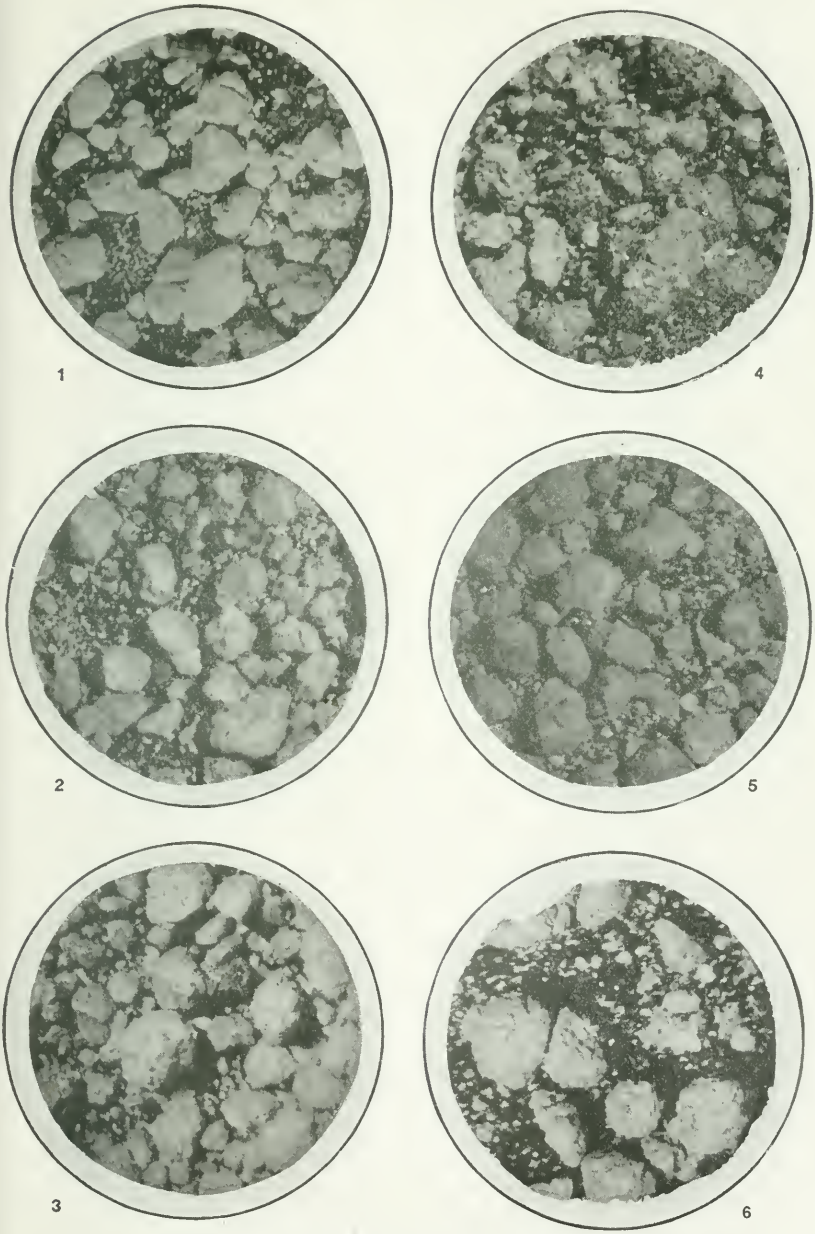
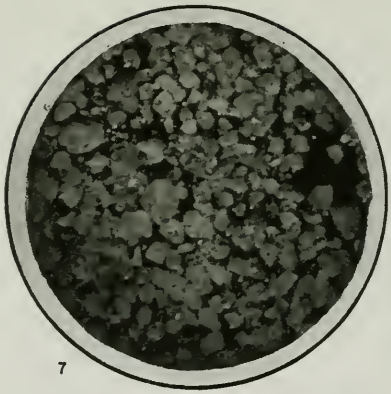
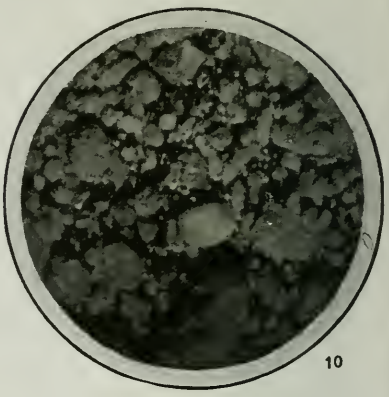
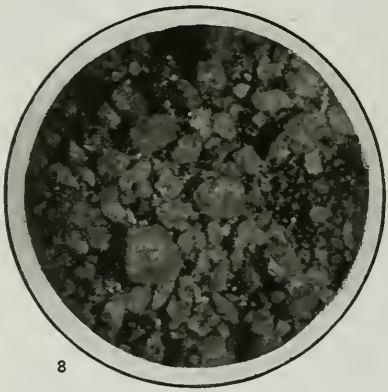


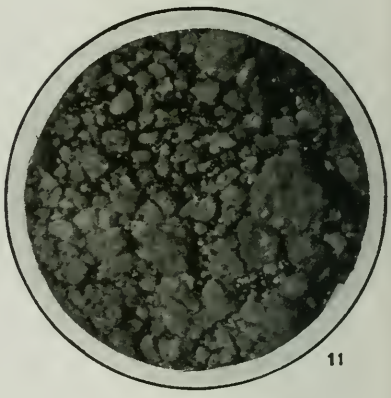
Plate I.—PHOTOMICROGRAPHS No. 3, ALBANY SAND.
(Oblique reflected light—Magnification, 20 diams).

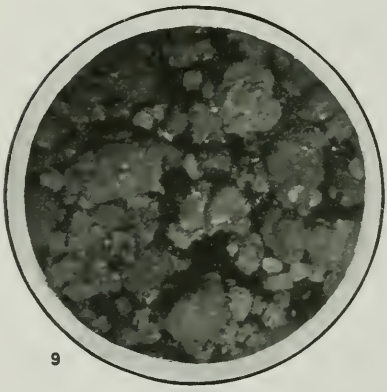
1. Fresh Sand.
2. After first burn.
3. After second burn.
4. After fourth burn.
5. After fifth burn.
6. After sixth burn.











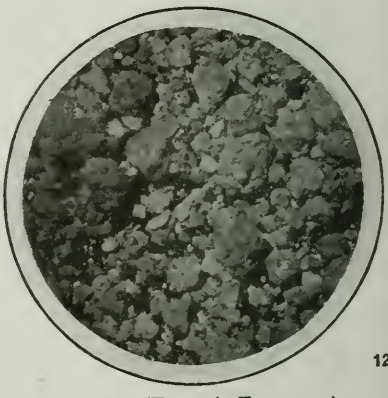
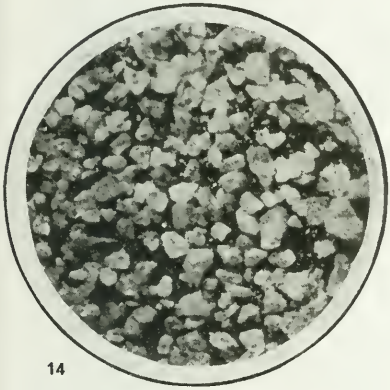


Plate II. PHOTOMICROGRAPHS BROCKVILLE SAND, (FLECK'S FOUNDRY).
(Oblique reflected light—Magnification, 20 diams).

7. Fresh Sand..
8. Afterfirst burn.
9. After second burn.
10. After third burn.
11. After fourth burn.
12. After fifth burn.



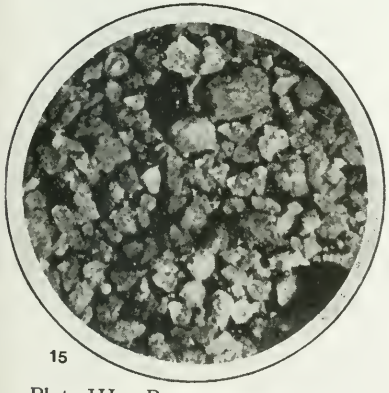


Plate III. PHOTOMICROGRAPHS BROCKVILLE SAND, (LAWSON'S FOUNDRY).
(Oblique reflected light, Magnification, 20 diams).

13. After first burn.
14. After third burn.
15. After fourth burn.
16. After fifth burn.
17. After seventh burn.
18. No. 0 Albany fresh sand.

taking the average of results obtained as the transverse strength, The results of the tests are given in Table No. 2.

Test for Refractoriness.—A test for the refractoriness of the fresh Brockville sand was made by preparing a cone and fusing it in an electric furnace with Standard Segars cones. The test cone fused at cone 8 which is equivalent to 1290 degrees C. or 2354 degrees F.

Microscopic Examination.—The fresh sand, as well as the samples from each burn, were examined under a binocular microscope, and the following notes made:—

1	No. 3 Albany fresh (*Fleck's*)		Small grains, well rounded. Larger grains, semi-angular. Quartz predominant. Occasional grains of magnetite—all were coated with clay, and of uniformly yellowish colour.
2	do	1st burn.	Quartz grains in some cases have the clay partially burned off. Sintering is seen in a few cases in the smaller grains. In most cases sand grains are unchanged.
3	do	2nd burn.	Similar to No. 2 with exception that the sand has a darker appearance, due to occasional grains having turned reddish in colour from the oxidation of the ferric iron coating.
4	do	3rd burn.	Sand has a darker appearance. Sintering more pronounced. Cementing of groups of the smaller grains together. Larger grains losing their clay coating to a small extent.
5	do	4th burn.	Similar to No. 4 with the grouping together of the smaller grains more pronounced.

6	Albany. (*Fleck's*)	5th burn.	Large grains have taken on the appearance of spongy masses due to the adhering of the smaller particles to them. Smaller particles are grouped and cemented together still further. Sintering quite pronounced. Occasionally edges of larger grains are fused.
7	Brockville fresh (*Fleck's Foundry*)		Sand consists of fairly clean quartz sharp and angular with only a very thin film of clay coating. Magnetite, hornblende, feldspar, and mica visible. Light yellowish colour.
8	do	1st burn.	Sintering appearing. Sand assuming a darker colour.
9	do	2nd burn.	Sintering quite pronounced. Larger grains have smaller grains cemented to them and the smaller grains commencing to group together in cemented masses.
10	do	3rd burn.	Sintering and cementing together of small particles more pronounced. Sand grains showing decided effect of heat.
11	do	4th burn.	Complete fusion in mass of some of the smaller groups of particles can be seen. The edges of most of the larger grains have become fused and present an appearance of water-worn sand.
12	do	5th burn.	Sintering pronounced in most of the grains. Small grains are grouped together and well fused. Larger grains show fusion of

			sharp edges. Bonding coating has disappeared from the greater percentage of grains.
13	*Lawson's Foundry*	1st burn.	Coating slightly affected. No sign of sintering. Sand similar in appearance to original.
14	do	3rd burn.	No sign of sintering. Coating only slightly affected. Otherwise similar to original.
15	do	4th burn.	Same as No. 14.
16	do	5th burn.	Same as No. 15 with slight signs of sintering.
17	do	7th burn.	Slight sintering. Smaller particles in some cases cemented together.
18	No. 0 Albany Fresh.		Highly silicious sand. Uniformly graded and coated. Sand particles sharp and angular. Light yellow in colour. Photomicrographs of each sample were taken, and are shown in Plates Nos. I, II, and III.

Chemical Analysis.—A chemical analysis of the Brockville sand was made, with the following result:—

Ultimate Analysis

S_2O^3	74.35	per cent.
Fe_2O_3	1.10	"
FeO	1.48	"
Al_2O_3	12.63	"
CaO	2.60	"
MgO	1.06	"
Na^2O	2.73	"
K_2O	2.15	"
H_2O	1.90	"

It is questionable whether much information can be obtained from ultimate analysis, beyond a very general indication of the refractoriness. It is moreover frequently very misleading; hence it is deemed advisable to omit it in future.

Permeability Test.—The permeability test shows that the permeability increases the oftener the sand is subjected to the molten metal. Results are given in Table III.

CONCLUSIONS TO BE DRAWN FROM RESULTS OF TESTS

An examination of the results obtained from the series of tests carried out on this sand brings out some interesting facts regarding the properties of moulding sands and their behaviour under actual working conditions.

Granulometric Analysis.—By comparing the No. 3 Albany screen analysis with the Brockville sand used at the Fleck Foundry, it will be seen that there is in both cases increase in the coarseness of the particles, the oftener the sand is submitted to the heat of the molten metal. In the case of the No. 3 Albany, this increase is uniform and very gradual; whereas with the Brockville sand the changes are abrupt and not uniform, showing a tendency of the smaller particles to become cemented together. On the other hand, the Brockville sand as used at the Lawson Foundry, shows only a slight increase after each burn, and the increase is uniform.

Tests for Transverse Strength.—In examining the results obtained by the tests for transverse strength it is seen that both in the No. 3 Albany and the Brockville sand used at Fleck's, there is a gradual decrease in the amount of water used and also a decided decrease in the transverse strength. In the case of the Brockville sand used at Lawson's the decrease in strength is only slight.

Microscopic Examination.—By microscopic examination of the samples obtained after each burn and also the fresh sand, some interesting data were secured. The Brockville sand (Fleck's),

shows decided sintering and cementing together of the smaller grains to the larger ones with a consequent decrease in bonding power. This sintering and cementing together of the particles is shown plainly in the photomicrographs, Plate II. The sand tested at Lawson's shows very little alteration when examined under the microscope.

Chemical Analysis.—The conclusions to be drawn from the chemical analysis are so slight that it can be dispensed with in an examination of this kind, the physical tests being the only ones of value.

SUMMARY OF RESULTS

Summing up the results obtained in the tests, it appears that the Brockville sand is a suitable moulding sand for stove plate and similar light work in iron, but although the heavier castings made in it were seemingly satisfactory it would not be advisable to use it on very heavy work, as the possibility of its failure would be greater than with the coarse sands in general use. Because of the fineness of its texture with the resultant tendency to sinter when exposed repeatedly to the molten metal, it appears to answer all requirements for use in the general run of brass foundry work. With a little care in selection and grading at the pit several grades uniform in texture could be obtained.

TABLE I

GRANULOMETRIC TESTS. CUMULATIVE PER CENTS.

Cumulative per cent of material retained on given meshes.

		6.	8.	10.	14.	20.	28.	35.	48.	65.	100.	150.	200.	200 +	Average Fineness
Brockville (Fleck's)	Green sand	.....	.....	.....	.....	.09	.24	.46	.69	1.63	8.85	49.95	77.21	.....	132.9
	1st. Burn	.....	.....	.27	.80	2.01	4.14	8.83	19.54	28.83	40.27	67.83	82.87	.....	102.0
	2nd. "	.....	.....	.18	.36	.78	2.00	4.88	12.00	16.95	28.80	60.27	80.79	.....	114.4
	3rd. "	.....	.....	.41	.89	2.01	3.91	8.84	20.18	27.70	41.96	68.35	83.00	.....	94.7
	4th. "	.....	.30	.80	1.26	2.17	4.46	8.68	19.70	29.03	41.23	70.26	85.12	.....	99.21
	5th. "	.....	.31	.74	1.41	2.46	5.15	10.98	24.60	34.05	48.09	72.46	80.02	.....	96.6
No. 3 Albany (Fleck's)	Green sand	.....	.....	.40	1.00	1.98	4.65	13.08	34.03	50.65	65.96	78.51	83.77	.....	81.3
	1st. Burn	.....	.....	.69	1.30	2.47	4.74	12.37	32.82	50.66	66.85	79.58	85.55	.....	79.7
	2nd. "	.....	.36	.91	1.56	2.76	5.55	13.72	36.09	52.11	68.03	79.65	86.01	.....	78.2
	3rd. "	.....	.40	1.03	1.86	3.52	6.68	16.13	36.52	54.18	69.33	82.22	92.34	.....	77.5
	4th. "	.42	.92	1.43	2.13	3.39	6.89	16.53	39.35	54.37	70.24	82.68	87.90	.....	73.5
	5th "	.....	.50	.90	1.63	2.89	6.23	15.68	37.71	52.85	68.96	81.30	87.78	.....	75.6
Brockville (Lawson's)	Green sand	.....	.....	.....	.....	.09	.24	.46	.69	1.63	8.85	49.95	77.21	.....	132.9
	1st. Burn	.....	.....	.....	.05	.18	.46	.75	1.22	2.79	11.56	48.43	74.05	.....	134.0
	3rd. "	.....	.....	.....	.13	.26	.44	.81	1.21	2.31	10.41	51.75	77.06	.....	131.3
	4th. "	.....	.....	.....	.11	.38	.69	1.95	1.42	2.87	11.44	46.25	72.39	.....	135.8
	5th. "	.....	.....	.....	.18	.38	.68	1.16	1 93	3 91	12.83	50.46	72.91	.....	132.7
	7th. "	.....	.....	.20	.40	.62	1.02	1.42	3.37	4.22	11.71	49.06	60.75	.....	140.1
No. 0 Albany Lawson's	Green sand	.....	.....	.14	.46	.88	1.38	2.03	3.01	5.03	11.27	19.46	26.64	.....	171.4

TABLE II

TRANSVERSE STRENGTH IN LBS.

(500 grams sand used in all tests; therefore, percentage of water used is double number of cc's. used divided by 10)

	H_2O used.	45 cc.	50 cc.	55 cc.	60 cc.	65 cc.	70 cc.	75 cc.	80 cc.	85 cc.	90 cc.	95 cc.	Average.
Brockville, Fleck's	Green sand	....	....	....	....	....	0.76	0.85	0.87	0.96	0.97	1.01	0.93
	1st. Burn	....	....	....	....	....	0.75	0.84	0.90	0.97	....	....	0.865
	2nd. "	....	....	....	....	....	0.86	0.85	0.86	0.81	....	....	0.845
	3rd. "	....	....	....	0.80	0.87	0.89	0.90	....	....	....	....	0.865
	4th. "	....	....	....	0.76	0.75	0.65	0.85	....	....	....	....	0.752
	5th. "	....	....	....	0.76	0.78	0.74	0.79	....	....	....	....	0.767
No. 3 Albany, Fleck's	Green sand	....	0.74	0.73	0.93	1.07	....	....	....	....	....	....	0.867
	1st. Burn	....	0.62	0.71	0.57	....	....	....	....	....	....	....	0.633
	2nd. "	....	0.57	0.69	0.65	0.74	....	....	....	....	....	....	0.665
	3rd. "	....	0.65	0.65	0.80	....	....	....	....	....	....	....	0.700
	4th. "	....	0.62	0.63	0.62	....	....	....	....	....	....	....	0.623
	5th. "	....	0.58	0.59	0.55	....	....	....	....	....	....	....	0.573
Brockville, Lawson's	Green sand	....	....	....	....	....	0.76	0.85	0.87	0.96	0.97	1.01	0.903
	1st. Burn	....	....	....	....	....	0.84	0.76	0.84	0.89	0.95	....	0.856
	3rd. "	....	....	....	....	....	0.92	0.97	0.96	0.85	....	....	0.875
	4th. "	....	....	....	....	....	0.90	0.92	0.88	0.88	0.85	....	0.886
	5th. "	....	....	....	....	....	0.91	0.82	0.97	0.88	....	....	0.895
	7th. "	....	....	....	....	....	0.85	0.81	0.84	0.89	....	....	0.847

TABLE III

Time in seconds. PERMEABILITY TEST. Gas Pressure = 2.8 inches.

		1st. Trial	2nd. Trial.	3rd. Trial.	Average.	
No. 3 Albany, Fleck's Foundry	Fresh,	4 1/5	5	4	4 2/5	
	1st. Burn	4	3	3 1/5	3 2/5	
	2nd. "	2 4/5	2 4/5	2 4/5	2 4/5	60 cc. water used = 12%.
	3rd. "	2 4/5	2 4/5	2 4/5	2 4/5	
	4th. "	3	3 1/5	2 1/5	2 4/5	
	5th. "	2 3/5	2 3/5	2 3/5	2 3/5	
Brockville Sand, Fleck's Foundry	Fresh,	19	17 3/5	18	18 1/5	
	1st. Burn,	8	7 3/5	8 2/5	8	
	2nd. "	8	8	9 1/5	8 2/5	70 cc. water used = 14%.
	3rd. "	5	6	7	6	
	4th. "	7	5 1/5	5 1/5	5 4/5	
	5th. "	5 1/5	6	6 3/5	5 1/5	
Brockville Sand, Lawson's Foundry	Fresh,	19	17 3/5	18	18 1/5	
	1st. Burn,	10 2/5	10	10 2/5	10 1/5	
	3rd. "	12 3/5	10 1/5	11	11 1/5	70 cc. water used = 14%.
	4th. "	8 1/5	8	8 1/5	8 1/5	
	5th. "	11	11 4/5	10 4/5	10 4/5	
	7th. "	8 4/5	7 3/5	6 4/5	7 4/5	
Albany No. 0	Fresh,	23	23	21 3/5	22 3/5	70 cc. water used = 14%.

POTASH, ITS PRESENT AND OTHER POSSIBLE SOURCES.[1]

By C. W. Drury

Uses of Potash.—Potash[2] is used mainly as an ingredient of fertilizers, about 90% of the total potash salts imported into United States being employed for this purpose, while the remaining 10% finds its way into the chemical industries in the preparation of paints, photographic supplies, dyes, fireworks, matches, and gunpowder.

Prior to 1887 more potash was used for industrial than for fertilizing purposes, but since that date its use as a fertilizer has increased rapidly. In 1890, 58.4% of the total consumption of German potash salts was for agricultural purposes; in 1900, 76.7%, and in 1909, 87.4%. In the United States the consumption of potash for fertilizer purposes practically doubled between 1904 and 1913.

In 1860 wood ashes were the chief source of potash, Russia being the leading producer.

Function of Potash.—Potassium is one of the three essential plant foods, the other two being nitrogen and phosphoric acid.

---

[1]In the preparation of this paper, the author obtained considerable material and information from the Report of the Fertilizer Industry by the Federal Trade Commission appointed by the United States Government, 1916; also from a Bulletin on Potash Salts, 1915, by W. C. Phalen, U.S. Geological Survey.

[2]The element potassium, represented by the symbol K, is the basis of all potash salts or compounds. This substance is a metal; that is it possesses metallic properties. To prevent rapid change it must be kept from air and water, with both of which it combines with great avidity. Combined with oxygen it forms potassium oxide, represented by the symbol K_2O, known as potassa, but popularly as potash. In estimating the quantity of potassium in the different products of the Stassfurt deposits, this compound, K_2O, is employed as a standard, the object being to establish a basis of comparison for all potassium salts. Among chemists as well as laymen there has grown up the practice of using for this standard the term potash. When only the term 'potash' is used in speaking of potash products, it is understood to refer to the potassium oxide (K_2O) present. As a matter of fact, however, potash salts are not sold in the form K_2O, but as the sulphate or the chloride. By the term 'potassium sulphate' is meant potassium (K) combined with the acid radicle of sulphuric acid (SO_4), or potassium oxide (K_2O) combined with sulphur trioxide (SO_3), making the compound K_2SO_4. By 'potassium chloride' is meant potassium (K) combined with another element, chlorine (Cl), and represented by the formula KCl.

It is especially necessary in the formation of starch, sugar, cellulose, and other carbohydrates. It also facilitates the growth of the fibrous woody portions of the roots, leaves and stems and has a beneficial effect upon the maturity of crops, the flavour and quality of fruits and the colour of flowers. Potash strengthens the straw or stalk and helps to fill the grain and fruit. In addition to the three essential plant foods, there are about ten others which are necessary for the normal growth and development of plants. Among these lime, magnesia, iron, soda, and silica are the most important. These latter plant foods, although important, are not usually added in fertilizers, it being assumed that the plant can extract sufficient from the soil.

Present Sources of Potash.—The main source of potash is the Stassfurt salts of Germany. The potash occurs in the form of the chloride or sulphate and the average grade of potash salts mined averages about 9% K_2O. A small quantity is obtained from wood ashes, tobacco stems, beet-sugar factories, alunite, kelp, brines, saline deposits, and silicate rocks. The four latter sources, especially the silicates, contain abundant supplies of putash, but the potash is present either in small quantities or else in a form not readily available as a plant food. The potash salts of Germany are practically all soluble in water and after several crystallizations a high-grade product may be obtained. Different grades of potash salts are produced and exported from Germany, the amounts being given in the following table. The German Government allows only about one-half of the output to be exported.

Production metric tons	Form of Salt[1]			
	Carnallite	Kainite	Muriate of Potash	Sulphate of Potash
1890	838,526	401,870	.....	.....
1895	782,944	669,531	.....	.....
1900	1,697,803	1,189,394	271,512	33,853
1905	2,239,709	2,405,536	373,177	47,994
1910	3,582,885	4,577,893	741,259	84,583
1913	5,302,350	6,305,160	506,744a	123,407a

[1]In the above table carnallite, kainite, muriate of potash and sulphate of potash contain 9%, 12.4%, 50.5% and 47-52.7% K_2O, respectively.

THE GERMAN POTASH INDUSTRY

Importance of the Industry.—The importance of potash in the agricultural world and the natural monopoly of production at present enjoyed by Germany has led to large investments of capital in the mining and refining industry. This has been especially the case since the enactment of the German law of 1910, which fixed the selling price of potash in such a manner as to insure a profit to the operator. The industry represented an investment of 500 million marks ($119,000,000) in 193 mines, (January 1915), with a considerable number of others in the course of development.

The Prussian Government owns mines at Stassfurt, Bleicherode, and Vienenburg; the Duchy of Anhalt has large works at Leopoldshall; and the Duchy of Brunswick is also interested financially in the industry. Most of the mines, however, are privately owned.

Extent of Deposits.—The principal crude potash salts mined in Germany are carnallite, kainite, sylvinit and hartsalz. The formula and composition of carnallite and kainite are as follows: carnallite, $KCl.\ Mg\ Cl_2 + 6H_2O = 16.9\%\ K_2O$; kainite, $MgSO_4$; $KCl + 3H_2O = 19.1\%\ K_2O$. These minerals contain muriate of potash, mixed with various other salts, especially sodium chloride, which are of little or no value for fertilizer purposes. The deposits of these potash minerals are found a thousand feet or more below the surface, immediately above the rock-salt deposits. The different grades of crude potash salts lie in layers, one above the other, the carnallite stratum being the thickest.

The potash beds underlie a large area around Stassfurt and the Harz Mountains. The known deposits of potash extend from Hecklenburg on the north to Thuringia on the south for a distance of about 140 miles; and from Hanover on the west to the Russian border on the east, roughly 250 miles. In 1904 important deposits of potash salts were also discovered in Alsace, while boring for oil was in progress, and it was found that the beds underlie nearly 7 square miles and range from 6 to 30 feet in thickness. Several shafts have been sunk here for mining the salts. Other deposits have been reported in Baden. It is

estimated that the total German deposits of potash are sufficient to supply all the requirements of agriculture for several centuries.

In no other country have so extensive and accessible deposits of potash salts been found as in Germany It is stated that there are large deposits of such salts in Russia, but they are too deep to be mined profitably. Some low-grade deposits have also been found in Austria-Hungary, and one mine located near Kalusz in Galicia has been operated. Recently, deposits have been found in Spain but it will be sometime before their extent and commercial value can be determined. Deposits, which may prove to be of importance, have also been found in India.

Growth of Industry.—Potash salts were first discovered in Germany in 1857 while borings for rock salt were being made at Stassfurt near the Harz Mountains. The value of these salts as a fertilizer was discovered in 1860, and the first factory for refining crude potash was erected in 1861. In that year only 2,300 metric tons of crude salts were taken from the mines.

Mining and Refining.—Some of the crude potash salts mined in Germany, especially kainite, containing 12% K_2O as mined, are ground to powdered form and used as a fertilizer without any refining. Carnallite is little used in this way except near the mines, because it absorbs water readily and is too expensive to transport. It is the principal source of muriate of potash and other refined salts. As mined, carnallite contains about 9% of actual potash. In making muriate it is ground and mixed in a dissolving vat with a boiling saturated solution of magnesium chloride. It is then concentrated until it shows 32° Be^1, when it is run into settling vats from which the clear solution is drawn off into crystallizing vats, cooled, and allowed to crystallize for several days. Manufactured muriate of potash is usually sold on the basis of 80% muriate or 50.5% actual potash.

Sulphate of potash and magnesia or double manure salt was formerly made from kainite, but now is made by mixing muriate of potash with a concentrated solution of kieserit, a raw salt containing sulphate of magnesia. It is sold on the basis of 48% of actual potash. Sulphate of potash is formed from the double sulphate of potash and magnesia by adding a certain quantity

of muriate. It varies from 90% to 96% pure sulphate, which is equivalent to from 47% to 52.7% actual potash. Single manure salt, now imported in two grades 20% and 30%, is a partly refined salt containing muriate of potash. Hartsalz contains about 16% potash and kainite about 12%. Sylvinit is very similar to hartsalz and contains 17.4% of potash.

Of the refined salts muriate of potash 50.5% and sulphate of potash 47% to 52.7% are used most extensively in the United States. Of the unrefined salts kainite is most generally used, though manure salt and hartsalz are imported in considerable quantities. No sylvinit has been imported since 1912.

Sulphate of potash, muriate of potash, double manure salt and 30% manure salt, all of which have been refined, are classed as concentrated forms and shipped to the United States in 200-pound bags, whereas 20% manure salt, although partly refined, hartsalz, and kainit, are classified as crude salts and are shipped in bulk.

The Potash Syndicate

History.—Although the deposits of potash salts in Germany were discovered in 1857, and the first muriate factory was established in 1861, it was not until about 1875 that the mining of potash salts began on an important scale. Between that year and 1879 four mines and several muriate factories began operations. Competition among them became so severe that in February, 1879, they entered into an agreement to regulate the price of carnalite for five years, to fix the maximum quantity of crude salts to be produced, and to determine the distribution among the factories. The states of Prussia and Anhalt operated potash mines and became members of this combination. In the following year, 1880, the agreement was extended to cover manure salts also. In 1883 the entrance of the Aschersleben Kaliwerke, a powerful independent mine, into the field, caused the dissolution of the first agreement and the formation in October of that year of another agreement for four years. This new agreement embraced all the mines and factories, and included the regulation of the price of kainite for export, and fixed the quota of the five mines then in operation. By 1888 two more mines

were operating, and the seven formed, for 10 years, a combination which apportioned the output of the mines and handled sales through a syndicate or selling company.

Shortly afterward a propaganda bureau and a statistical bureau were created for Germany by the syndicate and later on similar bureaus were established in foreign countries, their duties being the distribution of potash literature and the encouragement of experimental work among farmers. By 1907 there had been 41 of these bureaus established and prior to the outbreak of the European war in 1914, there were 60 in operation.

From 1890 to 1900 the number of active mines increased to 15, some of which had not entered the syndicate, and in 1901 and 1902 the competition between them and the syndicate in the American market became so severe that prices were cut in half, and the independents were forced into the syndicate.

The agreement which come to an end on June 30, 1901, was renewed for three years, but only at the last moment, on account of dissensions among the members over their respective quotas. The agreement which expired in 1904 was renewed with greater difficulty until June 30, 1909.

All of the potash mines did not join the combination during the period from 1904 to 1909, and the syndicate itself had difficulty in keeping members from secretly breaking their agreement. The total number of mines increased from 15 in 1900, to 69 in 1910, and dissensions among the members increased accordingly.

The syndicate agreement which expired in 1909, contained a provision to the effect that members would be free to make independent contracts after June 30, 1909, if arrangements for renewing the combination were not made prior to that date. Efforts to complete such arrangements were unsuccessful, and on the night of June 30, 1909, the owner of a controlling interest in two of the most important potash mines entered into contracts with an American agricultural company for the delivery of a large quantity of potash salts at much reduced prices. The contracts covered a period of two years, with the privilege of renewal for five years more, and called for deliveries of not less than 27,000 tons of potash (K_2O) annually. Subsequently these two mines made additional contracts with American buyers at

the same prices. Through these contracts the two mines secured control of about 90% of the total requirements of the United States for a period of seven years. The prices agreed upon in these contracts were from 25 to 38% less than the syndicate prices, and also even less than the actual cost of production of a number of mines. Muriate, for instance cost $15.54 and kainite $1.92 per net ton f.o.b. mine under the independent contracts, or $20.40 and $5.16, respectively, charged by the syndicate.

It was asserted that if the independent contracts were allowed to go into effect the weaker mines would be forced to close and the investment in them lost. At the same time the two independent mines, with the greatly increased production and consequent reduction of cost could operate profitably at these prices.

Immediately following the developments of June 30, 1909, negotiations were begun by the various mine owners looking to a renewal of the syndicate. A provisional syndicate was organized on July 24, which immediately set about to obtain a surrender of the independent contracts. The owner was offered a large sum to cancel them, but he refused and, as stated above, entered into further contracts with American buyers. The syndicate then sought the aid of the German Government, and on May 25, 1910, the federal law concerning the sale of potash went into effect. The requirements of this act placed such a burden upon the American buyers that the independent contracts were cancelled and the syndicate was renewed upon a firmer footing than was possible before. In making the contracts with the American buyers, the independent sellers of potash salts had inserted a clause in the contract to the effect that the American buyers should pay any governmental charges that may be imposed. The German government imposed a heavy tax on all production in excess of their regular allotments. The quantities of salts produced by the two independent mines were far in excess of their allotments, so that they were subject to heavy excess payments. As these excess taxes had to be paid by the American purchaser, the cost of potash salts with the added tax was in excess of the syndicate prices, so the contracts were cancelled. The two independent mines were forced to enter the syndicate, so that now it controls the entire output of potash salts by German mines

and fixes prices and terms of sale in every consuming country. In 1914 the life of the syndicate was extended to 1925.

German Kali Works.—The German Kali Works is the porpaganda agency of the German potash syndicate in the United States. It is incorporated under laws of the State of New York and has headquarters in New York City. This agency publishes and distributes literature concerning the use and importance of potash salts and the feasibility of home mixing for the farmer. It also co-operates with agricultural colleges and societies and with experimental stations for the purpose of increasing the the consumption of potash salts in the United States.

In addition to its propaganda work, the German Kali Works sells potash salts directly to the farmer, and also, from time to time, renders assistance to manufacturers who get their supplies from the syndicate but have need in an emergency of an immediate delivery. In like manner the German Kali Works itself purchases from the manufacturers when its needs can be more conveniently served by so doing. Prior to 1911 this agency also sold all kinds of fertilizing materials, but by the terms of the settlement of the potash controversy the Potash Syndicate agreed not to sell any fertilizer in the United States except potash salts until after the expiration of the five contracts at the end of 1916.

Owing to the manner in which potash salts are sold in this country by the German Potash Syndicate, the price of potash is not regulated by supply and demand as may be seen from the following paragraph, all prices as given are c.i.f. Atlantic ports.

The syndicate prices in January, 1910, were $32.98 net for muriate and $7.03 for kainite, while the corresponding independent prices at that time were $20.40 and $5.16, respectively. After the passage of the German potash law of 1910 the syndicate prices were raised to $35.75 per ton for muriate and $7.50 for kainite, and the shipments of potash under the independent contracts were subject to a heavy excess tax.

With the settlement of the potash controversy the list prices to the United States were increased, but the discounts were graded, so that for quantities in excess of 20,000 tons of potash (K_2O) they amounted to 15% on concentrated and 15.5% on

crude salts, bringing the net prices for large contracts below the net price in effect prior to April 30, 1911. On January 1, 1914, the list prices were again raised, while the discounts remained unchanged, so that the net prices on the large contracts were $33.21 per ton for muriate.

So bitter was the controversy of 1910 between the United States and Germany and so great was the desire and necessity to have an independent supply of potash, that the United States Government commenced extensive investigation work in search for a deposit of potash, or for a method which would render available the vast amounts of potash contained in the various sources that existed on this continent. Appropriations for this work have been made annually by the United States Government since 1910, and in August, 1916, the Congress of United States voted $175,000, for the investigation of various sources of potash within the United States. The work accomplished so far by the different departments of the United States Government tends to show that there are no large known deposits of potash salts on this continent similar to the Stassfurt deposits and that before potash can be recovered from the present known sources, considerable difficulties have to be overcome.

Before discussing the various sources of potash on this continent, a brief summary of deposits of potash salts in other countries is given. Although reports of large deposits have appeared in the journals frequently, it may be safely said that no deposits sufficient in size to compete with the Stassfurt salts have been discovered.

OTHER POTASH DEPOSITS, THE OCCURRENCE OF WHICH IS KNOWN OR HAS BEEN REPORTED

Austria.—In 1910 it was reported that potash salts in quantity had been discovered near Kalusz, in Galicia, Austria. No authoritative statement has appeared as to the extent of these deposits but it need hardly be expected that these deposits, even though they be large, will compete with the German potash salts. The deposits near Kalusz are of a somewhat different formation from those of Stassfurt. The annual production of the largest mine is

1,000 tons of potash which is only about one-half the consumption of Galicia itself.

Alsace-Lorraine.—In 1912 attention was drawn to the possibility of the existence of large deposits in Upper Alsace. A report dealing with the mineral production of Alsace-Lorraine states that in 1912 and 1913, there were produced 137,243 and 350,341 tons of potash respectively.

Potash deposits in India.—According to a report of the Geological Survey of India there is a possibility that potash deposits may be developed at Khewra and Nurpur in the Punjab salt range.

Potash deposits in Spain.—From time to time during the past few years consular reports have been made claiming the existence of potash deposits in the vicinity of Barcelona, Spain. Up to the present no potash in quantity has been produced from these deposits but it is reported extensive development work is being conducted by Belgian, French, German, and Spanish capitalists.

Potash deposits in South America.—Statements with analyses of the salts have appeared concerning the existence of potash in Peru and Chile. A report of the occurrence of large deposits in Cuba proved to be exaggerated. At present little information is available concerning deposits which may exist.

OTHER POSSIBLE SOURCES OF POTASH

Potash from wood ashes.—Wood ashes were originally the source of potash until the discovery of the Stassfurt deposits. Formerly Canadian wood ashes furnished the best grade and large amounts were marketed in the United States, a considerable tonnage going as far south as Virginia. Ordinary wood ashes contain 5-10% potash, as potassium carbonate, and used to sell in 1900 at $4 to $5 per ton, f.o.b. Lake Ontario ports. At the present time due to the scarcity of potash salts, ashes are selling for about $10 per ton but it is impossible to obtain any large quantity of wood ashes in Canada, owing to the decreased consumption of wood for fuel, and also to the fact that what wood ashes are

produced on farms are now applied directly to the soil by the farmers, who have realized its fertilizing value.

The production of wood ashes has been carried on in South Russia for some time but at present shipments are stopped. The shipments formerly amounted to 300 tons per month. In the Caucasus there are 24 factories producing potash from the stems of sunflowers. In 1907, 14,500 tons were produced.

The sawdust of the large saw-mills furnishes another large, possible source of potash. Immense quantities of sawdust are produced in the Western mills and the subject has been carefully reviewed by Bradley[1] and Zoller.[2]

The possibility of recovering potash from sawdust and mill waste is summarized as follows. Is it more profitable to burn the sawdust and extract the potash or to use the mill-waste for fuel, or for the production of soft wood cellulose from which it is possible to obtain a good grade of paper pulp, wood flour, and ethyl alcohol; or will the value of the extracted potash be equivalent to the loss of charcoal, wood alcohol, wood-oils and other by-products which would be lost in the gases from the incinerator?

Potash from Wool.—In the scouring and washing of wool, the wash water obtained contains about 3% of total solids which contain 25% potash. The potash is in the form of carbonate and potash soaps but the potash obtainable from this source cannot be very great. The recovery of potash salts from wool was undertaken quite extensively in France but the industry has practically been abandoned.

Potash from Fish Water.—Fish water produced in menhaden factories contains about 8% solids which averages 5% potash and 16% ammonia. The industry is not a large one and is confined to factories along the coasts. The estimated recovery from this source is about 800 tons of potash annually.

Potash from Distillery Waste.—The Potash Recovery Co. Sausalite, California, has been organized to recover potassium from distillery waste.

Potash from Sugar Residues.—In extracting sugar from cane or beets, a large proportion of the soluble mineral matter in the

[1]Metallurgical and Chemical Engineering Vol. 13, 1915 p. 841.
[2]Journal Industrial and Engineering Chemistry, Vol. 8, 1916, p. 105.

the cane or beet is extracted with the sugar juice and, upon crystallization of the sugar, remains in the molasses residue. Whether the molasses is used for fermentative purposes or the sugar recovered by means of barium, a final end-product is obtained containing the soluble potash, and this end-product contains, on an average, 4% solids containing approximately 10% potash in the form of carbonate. Taking the production of sugar beets during the years 1913 and 1914 as an average, the total amount recoverable would amount to about 25,000 tons of potash. However, it is safe to assert that beet sugar potash will always be insufficient to replace the potash used in growing sugar beets.

Potash from Copper Ores.—A large number of analyses and estimated tonnages have been published showing the amount of potash in copper and gold ores and tailings of the Western States. The potash in the ores and various tailings average about 5% K^2O but the potash exists in the form of silicate. Before it is possible to consider recovering potash from such a source, a satisfactory method to treat silicates must be devised.

Recovery of Potash in the Cement Industry.—The raw materials used in the cement industry contain small quantities of potash and at several plants, (viz., Riverside Portland Cement Co. Riverside, California; Security Cement and Lime Co., at Security, near Hagerstown, Md.; and the Universal Portland Cement Co., Duluth, Minn.,)methods have been installed to recover the potash from the flue gases. The Cottrell process of electrical precipitation is being used.

At the Riverside plant the potash present in the ordinary raw material charged into kilns amounted to approximately 0.25% K_2O but recently a higher grade raw material is being used. The potash content in the final dust recovered from the treating system, contains from 12 to 20% K_2O. The efficiency of collection of the suspended particles averages 96%, the escaping material being largely potassium and sodium sulphates. About 90% of the potash present in the raw material is volatilized. The amount of dust collected by the system averages 90 tons per 24 hours and from this about four to five tons of final product containing 20% K_2O is produced. A report states that the

Security Cement and Lime Co., will save about two tons of potash daily.

The Cottrell installation at Riverside treats about one million cubic feet of gas per minute, the capacity of the plant being about 5000 bbls. of cement per day.

It has been suggested that in the cement industry a large proportion of feldspar might be substituted for clay, and during the burning of cement the potash would be volatilized. But there are practical obstacles to the carrying out of this proposal, including the difficulty in many cases of procuring the feldspar, the greater cost of feldspar than clay, the extra cost of grinding, and also the extra freight, since abundant supplies of feldspar are not likely to be situated near cement mills. There is also the installation of the precipitating or recovery apparatus. In the case of the Riverside Cement Co. installing the Cottrell system to recover the flue dust escaping into the air, it was necessary that some system be installed to recover the dust because previous to the installation damages aggregating large sums had been paid by the cement company, and so strong was the feeling of the orange growers in the vicinity of the cement plant that the factory was threatened with an ultimate injunction.

Besides the Cottrell system, bag-house and spraying methods have also been mentioned, but, up to the present time, the Cottrell is the only successful recovery process. The cost of recovering potash at the Riverside plant has been estimated at 34 cents per unit of potash.

Recovery of Potash from Blast Furnace Gases.—From time to time statements have appeared regarding the enormous loss of potash in connection with the operation of blast furnaces. An extensive investigation of the question of recovering potash from blast-furnace gas has been carefully conducted at the Bethlehem Steel Works. The results of these tests have been published recently by Wysor[1] from whose article the following table and summary is taken. "Average iron ore contains approximately 0.25% K_2O; however, in the case of manganese ores the potash usually runs higher. Coke contains about the same percentage,

[1]Bulletin, American Institute of Mining Engineers, January, 1917

while certain samples of limestone contain more. The following table shows the behaviour of potash in the operation of the blast furnace and in the following paragraphs is given a summary of the possibility of recovering potash from blast-furnace gases.

	Potash in pounds per ton pig iron of produced	Percent of total charge
Total charged	22.4	....
Lost in slag	4.5	20.0
Lost in fume, etc., from iron and cinder notches, and shell (est. 10 % of slag loss)	0.4	2.0
Lost in gases from top of furnace (est. 5 % total of gas losses)	0.9	3.9
Lost by combination with brickwork of furnace, stoves, etc	negligible	
Recovered in dust-catcher dust	0.6	2.7
Lost in primary washers (est. by difference)	12.5	55.9
Lost in secondary washers	0.5	2.2
Recovered in stove and boiler passes	0.3	1.3
Wastage from mains, stoves (est. 30 % of quantity recovered)	0.1	0.4
Lost in stack gases	2.6	11.2
Total	22.4	100.0

In the present gas-cleaning practice it appears that there is a loss in the primary washers alone of over one half of the total potash charged, or about 12 lb. per ton of pig iron produced, though, as mentioned later, this amount is probably a little high. The amount recovered, while appreciable, and representing almost clear profit, is seen to be an insignificant part of the total—less than 2%. Potash lost in the slag, around the shell, and from the top of the furnace, for all practicnl purposes, is lost beyond recovery. The greater part of the alkali content of the flue dust removed from the dust catchers could be recovered by leaching in water, but the percentage is too low to justify reclamation in this way. However, the potash now lost in wet washers and from stove and boiler-house stacks offers a legitimate and inviting field for its recovery. According to the balance sheet, it appears that about two-thirds of the total potash charged is now lost in the wash

water and stack gases, or about 15 pound per ton of pig iron produced.

At intervals during the course of the last year or more, an experimental Cottrell electric dust precipitator has been in operation, being connected to the raw gas-main leaving one of the dust catchers. It is stated that practically all the dust and fumes entering the treater could be precipitated successfully. The colour of the dust recovered varied from a light to a dark gray. Several samples were analyzed and showed a potash content of about 10%. The total dust leaving the dust-catcher is evidently very much richer in potash than the relatively heavy particles constituting the dust in the dust catchers.

It may be mentioned that by weak acid treatment part of the insoluble potash content in flue dust may be rendered water-soluble though this is not likely to be of practical application. Also, the soluble alkali salts can be recovered in tolerably pure form by leaching and evaporation.

Potash from Alunite.—Large deposits of alunite, a hydrated sulphate of potash and alumina, occur near Marysville in the State of Utah, while smaller deposits are found in Nevada, and on Vancouver Island, British Columbia. The alunite of Nevada and Vancouver Island contains considerably more silica and less potash than the deposits of Utah which average 10% K_2O. Deposits of alunite also occur in Italy, Spain, Australia, and Japan. The Italian deposits have long been worked for the production of Roman alum. The deposits of Spain have also been worked on a small scale to produce potassium sulphate.

At present there are two companies operating in Utah to recover potash from alunite. The Mineral Products Corporation in which the Armour Fertilizer Co. and the United States Smelting and Refining Co. are interested, and the Utah Potash Company. The first carload of high-grade potassium sulphate (95% pure) was shipped by the Mineral Products Corporation, on October 20, 1915, and although the capacity of the plant is rated at 25-30 tons of potassium sulphate per day, no figures showing the actual production have yet been made public, the only statement being that the output of the plant has been contracted for.

Recovering potash from alunite appears simple, but no

doubt there are certain difficulties in connection with the process. To obtain a thorough decomposition of the mineral and a high extraction of potassium sulphate has presumably presented some difficulties.

Besides any operating difficulties to recover the potassium sulphate, some means have to be provided to prevent the oxides of sulphur from escaping into the air since the alunite deposits are located in a forest reserve, and there is practically no market for sulphuric acid in this locality.

A patent has been granted to Charles H. MacDowell, vice-president of the Mineral Products Corporation, to volatilize the the potassium sulphate by heating the roasted alunite with carbon to temperatures of about 800°C. The process used by the Utah Potash Company consists in mixing lime with ground alunite before roasting. The sulphur dioxide produced during roasting combines with the lime forming calcium sulphate.

It has been stated that the recovery of alumina from alunite should be a source of considerable revenue, but up to the present no large amounts of alumina, if any, have been sold for the production of aluminum. In the aluminum industry the requirements call for a raw material containing only 1% impurities divided between SiO_2, Fe_2O_3, TiO_2, Na_2O, and H_2O. The alumina by-product would be sold to replace the present raw material, bauxite, a hydrated aluminum oxide. The average price of bauxite[1] in 1910 was $4.89 per ton. Since one ton of alunite contains 740 lb. of alumina and most likely some sulphate, at the figure for 1910, it would be worth not more than $3.00. That there is some difficulty in finding a market for the alumina may be gathered from a recent report of the company which states that research work is being conducted to produce a highly refractory material from one of the by-products of the process. This by-product is most likely alumina.

From an economic standpoint it would seem more practical to use the ignited alunite directly as a fertilizer since the expense of separating the soluble salts from the alumina would probably be more than the value of the alumina obtained, but as California

[1]Bauxite contains when pure 73.9% Al_2O_3 and 26.1% water.

is the nearest and largest market for fertilizers, there would still be a long rail haul to dispose of the product.

Potash from Kelp.—Among the considerable number of sea algae on the Pacific coast occur four species which are peculiar to that coast and which have the power of absorbing from the sea water exceptionally large quantities of potassium. Of these four algae or giant kelps, as they are known locally, *Macrocystis pyifera* is the most important. The kelp beds along the Pacific coast aggregate 400 square miles or more in area, the largest areas being along the coasts of Southern California and Southeastern Alaska. The average length of the full-grown plant is about 100 feet. Beds of kelp also occur along the coasts of British Columbia, the Philippine Islands, Ireland, Scotland, and Japan. Green kelp as harvested contains approximately 85% water and 1.75% potassium oxide, also small amounts of nitrogen and phosphoric acid.

Three or four companies have already been formed to produce potash from kelp but up to the present no potash of importance has been marketed. Some companies merely dry and crush the kelp and sell it directly as fertilizer.

The question of harvesting and drying green kelp has presented many difficulties. It is stated that the kelp beds may be cut twice a year but in order to produce one ton of potassium chloride, 50% K_2O, fifty to seventy tons of green kelp would have to be harvested. The cost of harvesting green kelp has been given as 20 cents per ton. No reliable costs have been published showing the cost of drying green kelp. Air drying cannot be practised in the vicinity of the kelp beds on account of climatic conditions and the large areas required, so it is necessary to employ artificial means. Revolving furnaces seem to be favoured for the reason that the kelp must be constantly turned or a thorough drying is not obtained. An attempt was made to remove the excess water by filtration but this was not successful because the fibreless character of the organic matter clogs the filters. A British patent covers a process to destroy the pulpy character of the pulp by heating it to 170°C. This process does not appear to be practicable. Some advocates of the potash-from-kelp industry claim that iodine could be recovered as a

by-product, but it must be remembered that the present consumption of iodine is only about one-tenth the possible production of Chile. Japan also exports iodine. Up to 1885, Japan imported potash and iodine; since then she has extracted them from kelp and at present sends potash to Europe and iodine valued at $500,000 annually to the United States.

It may be of interest to note that the kelp industry once flourished in Scotland and was intimately connected with that of glass. It is stated the Celtic vitrified forts, which have puzzled archaeologists, were built in this way. Ramparts of sand were covered with dary seaweed and the seaweed set on fire. On the vitrified layer thus formed, more sand was piled and the operation repeated until a vitreous mass of considerable height and hardness was obtained. One of these forts may be seen not far from Connell Ferry in Argyllshire. In other cases the seaweed, dried as much as the climate would permit, was burnt in the open and the charcoal leached; considerable loss of salts resulted. As early as 1767, Delaval (Pat. 870) proposed burning the wet weed in reverberatory kilns by the aid of coal or wood. There were also one or two patents modelled on Le Blanc's soda process. In 1861 (Eng. Pat. 1411) in place of burning seaweed in the open, Stanford dried and pressed it into cakes, and destructively distilled it. The charcoal was withdrawn into closely-covered iron boxes to cool, leached and treated as usual for potash and iodides. The distillate gave gas and a tar liquid which yielded ammonia and other chemicals.

Gignon and Cagnage (Eng. Pat. 2741, 1865) proposed to disintegrate and press the kelp[1] preferably while wet, to extract the salts and nitrogenous organic matter. They steeped the residual pulp in a solution of sodium carbonate, and again pressed it to remove the greater part of the lye, which they used for the manufacture of paper.

After this potash became of secondary importance and iodides primarily were sought. In 1881 (Eng. Pat. 142) Stanford leached the kelp with hot or cold water, evaporated the liquor, and carbonized the extract for potash and iodine. The residual pulp after being boiled with alkali gave algin, which had only a

[1]Cresswell, Journal, Society of Chemical Industry Vol. XXXIV, 1915, p. 389.

moderate success. Soon afterwards even the iodine ceased to be profitable and the works were at last closed.

Recently the Board of Agriculture of Scotland, 29 St. Andrew Square, Edinburgh, has been taking steps to increase the output of kelp.

Boberg and Testrup (Eng. Pat. 1766, 1914), mince and pulp kelp, and heat in closed vessels to 170° C. for 20 minutes. The mass now yields a firm press cake. The liquor contains 70% of the original alkali salts, 90% of the iodine, 33% of the nitrogen, and 10% of the organic matter.

The Canadian Potash and Algin Co. Limited, has established a plant at Sydney, British Columbia. This company proposes to extract potash from kelp and to recover iodine and algin as by-products. The chief use of algin is in the sizing of fabrics.

The kelp beds along the Pacific coast all lie within the one marine league limit, hence control of them is vested in the States and not in the Federal Government and there does not appear to be any State law applicable. Capital fears competition, and competition, even to the harvesting of the same bed by rival interests, is quite possible now, legally as well as physically. This has been, and is yet, the principal difficulty in the development of a potash-from-kelp industry.

It is also to be noted that the market for potash is chiefly along the Atlantic Coast and the freight on potash salts from California to Atlantic ports would be about $2 per ton—about the same as the freight from Hamburg to Atlantic ports.

RECOVERY OF POTASH FROM SALT BEDS AND BRINES

Potash bearing salt beds occur in California and Texas; while the most promising brine lakes found are in Nebraska and Oregon.

Of the known salt beds the desert basins of Searles Lake, California, appear to be the most valuable. The salt beds contain brine solutions which contain a mixture of potassium and sodium sulphates, carbonates, chlorides, and borates. The brine solutions contain approximately 30% solids of which about 5% is potassium chloride. The exposed surface area of Searles

Lake is about 11 or 12 square miles and has an average depth of 60 feet. In order to obtain a ton of potash, about 100 tons of brine and salts will have to be treated and from the treatment a little more than one ton of anhydrous borax will be produced.

Already three companies, the California Trona Co., The American Trona Co., and the Foreign Mines Development Co., have erected plants to recover the potash salts from the brines of Searles Lake. The three companies mentioned are said to be closely associated with one another and with the Consolidated Gold Fields Co., of South Africa. The American Trona Co. has erected two plants, one at Searles Lake or Trona, as the place is now called, and the other at San Pedro, near Los Angeles. The plant at Trona will produce the concentrated salts which are to be refined at the San Pedro plant. The American Trona Co., had invested up to December, 1915, approximately $2,000,000, a large part of which was used to construct a railway 30 miles long.

Potash in Texas.—Considerable drilling and exploration work in search of potash has been done in Dickens, Potter, and Randall counties, Texas. The potash bearing material found is solid and not a brine. Drilling to depths of 2000 feet has been done; and samples taken from depths varying from 800 to 1700 feet show potash to be present in quantities from 0.31 to 10.5%. A potash content of less than 1% seems to represent the average content of the main salt beds of this region.

Owen's Lake, California.—Owen's lake is a large shallow body of water having an area of 97 square miles and a maximum depth of 30 feet. The water which is a dense brine contains common salt, soda, borax, and other chlorides in solution. The lake has no outlet and the volume of water in it depends upon the balance between the evaporation and inflow. The brine contains about 8% of solids of which 4% is potassium chloride. Crude carbonate of soda has been recovered from the brines of Owen's lake for the past 30 years by the Inyo Development Co. This company produced about a carload of potash salts in 1915 and is planning to enlarge the plant to recover the potash salts in quantity.

Great Salt Lake, Utah.—The Utah Chemical Co. has erected an experimental plant on the shores of Great Salt lake for ex-

tracting potash salts from the bitterns obtained as a by-product in the manufacture of salt by the Inland Crystal Salt Co. The company expected to produce some potash salts during 1916, but up to the present no large quantities have been shipped.

Potash salts from Nebraska Lakes.—There are many alkaline lakes in the Sand Hills region of Nebraska, the main one being Jesse lake situated in the extreme western part of the district. The water of these lakes is retained by an impervious layer of hard-pan or mud. The large deep bodies of water are only slightly alkaline, but some of the smaller ones are comparatively rich in salts. A large number of samples of muds, brines, and saline crusts of these alkaline lakes were taken and analyzed by the United States Geological Survey in 1914. The potash content of the various samples were found to vary greatly.

Of about 125 different samples taken, 87 contained less than 2% potash, seven contained between 2 and 3% while the remainder showed larger quantities. Analysis showed the mud to contain an average of 4.69% of soluble salts containing 19.86% K_2O. The analysis of the brine samples showed an average of 7.69% of dissolved salts containing 24.9% K_2O. The saline residues around the lakes contained on an average 19.5% soluble salts of which 14.7% was potash.

A small plant has been erected to recover the potash from the waters of Jesse lake. No attempt has been made to separate the pure potash salts, the final products shipped being crystals containing 30 to 40% brine, and brine solutions. Owing to the small area of the alkaline lakes, no great amount of potash salts will be produced from this source.

Saline Lakes or Deposits of Oregon.—Alkali lakes in Southern Oregon were examined by a member of the staff of the United States Geological Survey, but the results of the examination were not very encouraging. A sample of brine from Little Alkali, which is probably the largest body of water, contained 10.6% of dissolved solids of which only 6% was potassium chloride.

An examination was also made on the brines of Albert and Summer lakes, Oregon, but the analyses did not show the waters to contain any more potash than that present in sea water. It is

reported that the Oregon State land board granted a lease of three lakes for 40 years, the lessee agreeing to pay a royalty of $1 per ton on all soda and potash taken from the lakes and 50 cents per ton on common table-salt. Transportation is difficult since the nearest railroad is 40 miles from the lakes.

Salduro Salt Marsh, Utah.—A number of shallow holes were bored through the hard surface salt crust that forms the floor of this basin and into the underlying mud. Analyses of samples taken from these borings show that the surface salt is comparatively pure sodium chloride containing little potassium.

The mud contains about 12 to 15% of soluble salts analyzing 2.2 to 4.5% K_2O. Brine was found both in the salt crust and in the underlying mud and the analysis proved the brines to be saturated solutions carrying 25.8% and 26.8% of dissolved salts, of which 3.5 to 5% was potash expressed as K_2O.

The region around the Virgin River salt deposits, Nevada, was also examined but no definite deposits of potash were found.

The Railroad Valley Company has spent large sums of money prospecting for potash in Railroad Valley, Nevada. After extensive and earnest work this company is reported to have given up its search for potash.

POTASH FROM SILICATE ROCKS

Before referring to the silicate rocks as a source of potash the following table showing the composition of the various potash rocks is given.

FORMULA AND COMPOSITION OF POTASH MINERALS COMPOSITION (PER CENT)

Mineral	Chemical Formula	Silica	Alumina	Potash	Sulphur trioxide	Water
Leucite	$K_2O.Al_2O_3.4SiO_2$	55.0	23.5	21.5		
Orthoclase	$K_2O.Al_2O_3.6SiO_2$	64.7	18.4	16.9		
Muscovite	$K_2O.Al_2O_3.6SiO_2.2H_2O$	45.2	38.5	11.8		4.5
Alunite	$K_2O.3Al_2O_3.4SO_3$.$6H_2O$		37.0	11.4	38.6	13.0

Sericite—A variety of muscovite or common mica.

Glauconite—Essentially a hydrous silicate of iron and potassium.

Greensand—A mixture of glauconite, potassium and calcium phosphate, as well as calcium carbonate. The average potash content varies from 4-5%. The greensand bearing formation extends across sections of New Jersey, Delaware, and Maryland.

Potash from Leucite.—The mineral leucite occurs in large quantities in the State of Wyoming, and also in Europe. The mineral resembles orthoclase in composition but is more readily decomposed. When ground leucite is treated with sulphuric and hydrochloric acids complete solution of the bases result. It was thought that this process might prove a simple solution of the problem of recovering potash but it was found the resulting sulphates were about as hard to separate as the constituents of the original rock. Other wet methods were also tried such as treating the leucite rock with caustic lime under pressure, also using caustic soda and salt with the idea that the sodium might replace the potassium more effectively than lime. The results of these tests, however, were not encouraging.

Another method was to treat the mineral with sulphur dioxide gas at a temperature of 1,200°C. The Hargraves process, which is used for making sodium sulphate from common salt by passing steam and sulphur dioxide over it at a temperature of about 500°C., was also employed. The results from these tests were likewise not encouraging and for the time being experiments by wet methods have been discontinued.

Attempts to decompose leucite by dry methods have also been tried but without success. The following methods may be mentioned as summarizing the work done. Mixtures of leucite, lime, and gypsum were heated and the potash extracted in the form of sulphate. Attempts to volatilize the potash were also unsuccessful since the degree of volatilization could not be regulated. The Cowles process devised to extract alumina from clay, was also tried. In this process, a mixture of salt and clay is compressed into briquettes and heated, whereby hydrochloric acid is evolved and the alkali-silico-aluminate formed. This

compound is then decomposed by heating with limestone to form lime silicate, a soluble alkali aluminate being left. Briquettes were made and heated in the Cowles furnace but very unsatisfactory results were obtained. Succhell proposed heating leucite with potassium sulphate and calcium carbonate, in the proportions of 4:4:2. After the treatment the potash is present as a soluble silicate, the lime being changed to calcium sulphate.

Potash from Wyomingite.—Wyomingite, a lava occuring extensively in the Leucite Hills, in Sweetwater County, Wyoming, is composed largely of the mineral leucite. The composition of wyomingite has been given as follows: Uncombined silica, 22.5%; leucite 35.7%; phlogopite, 22.3%; diopside 10.7%; and accessory minerals 8.8%. An analysis of a sample of wyomingite showed 11.1% potash.

Various laboratory methods have been tried to recover the potash from wyomingite but no satisfactory method has been devised. The methods tried consisted in treating the mineral with the following reagents: gypsum and sodium chloride, sulphuric acid, potassium bisulphate, roasting with alunite (a hydrated sulphate of potash and alumina), heating with calcium carbonate, heating with calcium chloride, heating with magnesium chloride, heating with a bittern residue (containing sodium and magnesium chloride), and heating with ammonium sulphate. The tests made using alunite and calcium chloride gave the best results, the extraction depending on the amount of the reacting substances present. In the case of the treatment with alunite the best extraction of the potash of wyomingite was 55%, while by the treatment with calcium chloride an extraction of 73% was obtained.

Use of Phonolite as a Source of Potash.—Phonolite is a volcanic eruptive rock composed of sanidine, a variety of orthocalse; nephelite, a potash soda silicate; and hornblende. Various attempts have been made in Europe, especially in Germany, to use ground phonolite as a fertilizer. The results obtained show phonolite to possess a fertilizing value but the effect is considerably less than that produced by the Stassfurt potash salts.

Potash from Feldspar.—As a source of potash on this continent, the mineral feldspar offers many possibilities. It contains

when pure 16.9% K_2O, 18% alumina, and 65% silica, but the potash is so combined with the alumina and silica that it resists decomposition by any of the common acids or any simple treatment. Although experimental work has been conducted on the recovery of potash from feldspar for 70 years or more, at the present time there is no satisfactory process in operation. Owing to its insolubility in the common acids and its slow decomposition in the soil, feldspar, even when finely ground, is of practically no value as a fertilizer. If the potash in feldspar is to be of any value it must be changed into a soluble or available form.

Different methods have been proposed to decompose feldspar and recover the potash. These may be classified into three groups: First, those methods relying on natural agencies to decompose feldspar; second, wet chemical methods; third, dry reaction processes by which the potash is rendered soluble or available.

There have been no practical methods worked out to decompose feldspar by natural agencies or by wet chemical methods, the reasons being, either the time of treatment is too long or the extent of the decomposition was too low for the consumption and cost of chemicals employed.

It seems, however, as if some success has been attained in decomposing feldspar at high temperatures with the addition of chemicals. Most of these methods are based on the J. Lawrence Smith analytical method of determining alkalies in silicate rocks. In the analytical method of determining alkalies, feldspar, calcium carbonate, and ammonium chloride are mixed in the proportions of 1:6:1, and mixture heated at a low red heat. The alkalies, potash and soda, are changed to chlorides which are soluble in water. In the practical application of this analytical process, calcium chloride is substituted for the expensive ammonium chloride. In this connection two processes, using lime or limestone and calcium chloride may be mentioned, viz., the Cushman and Coggeshall, and the Brown processes. In the process of Cushman and Coggeshall[1], finely ground feldspar is

[1]Original communications, Eighth International Congress of Applied Chemistry, vol. 5, pp. 33-49, 1912.

[1]Original communications, Eighth International Congress of Applied Chemistry, vol. 15, p. 222, 1912.

mixed with 20% each of lime and calcium chloride, the mixture heated to approximately 1050°C. at which temperature the chloride of the calcium chloride reacts with alkalies giving sodium and potassium chlorides which are separated from the calcium-aluminium silicates by leaching with water. In this process the temperature is not sufficient to fuse the mixture.

In the Brown process which was to be operated at Buffalo N.Y., feldspar was mixed with lime or limestone, calcium chloride, and coke, and the mixture melted in a blast furnace. The potassium and sodium chlorides formed, being volatile at the temperature of the furnace, are collected by spraying the gases with water or by dry methods. In case of spraying methods, a solution of the chlorides is formed and this solution has to be evaporated to recover the potash salts. It is proposed to grind the melted slag formed which contains calcium and aluminum silicates and use it for Portland cement. The proportions of chemicals used are regulated to produce a cement or slag containing 55% lime.

In summarizing these two methods it might first be mentioned the first process is subject to a high initial cost in grinding and mixing the raw materials and second, both processes use the same by-product, calcium chloride. This chemical will certainly advance in price if an extended market is opened. In a number of experiments conducted to test these processes, it was found that in order to obtain a satisfactory decomposition of the feldspar, considerably larger proportions of calcium chloride were necessary than the amounts necessary to combine with the alkalies. The use of a larger amount than the theoretical may be expected but when operating on a large scale the consumption of calcium chloride may be reduced. It is interesting to note that an experimental plant has been erected to test the Cushman and Coggeshall process and it is hoped the experiments will be successful.

In considering the Brown process it is certain that considerable difficulty will be encountered in recovering the potassium chloride in the furnace gases. The present spraying towers used in connection with blast-furnaces do not recover all the soluble potash salts passed through them. To recover potassium chloride from furnace gases some modification of the present system of

treating gases will have to be devised. There is certain to be some difficulty in introducing the new slag product for cement, as competition with the present cement manufacturers will have to be overcome. The preparation of tile and paving brick has been also suggested as a possible use for the slag.

THE DRURY PROCESS

After spending a number of years studying the potash-from-feldspar problem the writer concluded that it did not appear practicable to extract potash from feldspar in water-soluble form and produce a high-grade potash salt. Also that the marketing of by-products from any treatment was an uncertain problem due either to limited markets or to entering an already well-established field. With these conclusions in mind, the following process was devised and which appears to possess certain advantages. In the Drury process, crushed feldspar, limestone, iron ore, and coke are charged into a blast-furnace. The mixture melts and is tapped from the furnace as a slag which, when ground, is suitable for fertilizing purposes. The potash, though not water-soluble, is set free by very dilute acids, even by the strength of the acids in the soil. Agricultural chemists are agreed that any material which is soluble in a 2% citric acid solution is available as a plant food. The feldspar fertilizer material made by the Drury process and which consists of a mixture of potassium, lime, aluminum, and iron silicates is all soluble in a 1% citric acid solution and therefore the potash is in a form readily available as a source of potash for fertilizer purposes. During the past two years field tests have been conducted to test the value of the feldspar potash fertilizer and satisfactory results have been obtained. It has been argued by many that unless potash is water soluble, it is not suitable for fertilizing purposes. In this connection it should be mentioned that when water-soluble potash salts are added to the soil they form potash compounds insoluble in water with the ingredients of the soil and this accounts for the potash not being washed below the roots of the plants. Further, it will be noted that the feldspar-potash product made by the Drury process is more soluble than basic phosphate slag, the value of which is determined by the amount of

phosphate soluble in a 2% citric acid solution. After considerable opposition the value allowed for the citric soluble phosphoric acid of basic slag is practically the same as that allowed for water-soluble phosphoric acid of acid phosphate. It may be of interest to know that basic phosphate slag is the principal source of phosphoric acid in fertilizers used in Europe, Germany alone in 1913 using more than two million tons. From the field tests that have been conducted and also from the fact that all the citric-soluble basic phosphate slag produced finds a market at a price about equal to the water soluble acid phosphate, there can be no doubt as to the suitability of the Drury feldspar-potash product as a plant food.

CONCLUSIONS

In concluding this review, the writer regrets he is unable to mention a number of other processes that have been proposed to recover potash, especially those devised to treat silicate rock. However, it may be stated that practically all possible deposits of potash salts and brines have been examined but no high-grade, extensive deposits of potash salts have been found in sufficient quantities to supply any large proportion of the quantity consumed on this continent. A number of plants will no doubt be operated to supply local markets but anything further need hardly be expected.

Whether it is possible to recover potash from kelp or not on this continent, opinion is divided. However, before extensive sums are invested it would be advisable to study the processes used in those countries where the operation was practiced or is being practiced, as in Ireland, Scotland or Japan.

The possibility of recovering potash from alumite presents certain difficulties which were mentioned in discussing the recovery of potash from alunite, but the investigation of the alunite deposits is being conducted by a company strong financially and there need be no fear but that the present company will make a success of the process if success is at all possible.

After following closely the different processes that have been proposed to treat silicate rocks, and also after a number of years of careful study of the problem, the writer came to the conclusion

that potash cannot be extracted from silicate rocks in a water-soluble form, and marketed under normal conditions; that very little can be realized by marketing alumina or silica as by-products; that the methods which aim to volatilize potash compounds and produce a residue suitable for cement, present difficulties such as the complete volatilization and collection of the volatilized alkali salts, the ability to obtain an easy running slag which will be suitable for cement, and further there will be the difficulty of introducing the products in a well-established market. It appears to the writer that those processes which merely treat the mineral so that the potash is rendered either water-soluble or available as a plant food, offer the best possibilities. Usually in such processes, other plant foods, such as lime, iron, soda, magnesia and silica, which should increase the value of the product as a fertilizer above the potash content are rendered available.

We still have the question of obtaining potash before us. We need potash and the need of potash is increasing annually. Experience has shown that the right use of fertilizers, proper tillage, good seed, crop rotation, and liming when necessary insure farm prosperity. Germany holds the present potash supply; what the price will be after shipping is resumed, it is difficult to estimate but it is quite safe to assert that potash will not be sold after the war at the same price that it was previous to 1914. Nevertheless the present crisis has caused practically every country to investigate their possible sources of potash salts.

———————

MINING METHODS AT THE MAGPIE IRON MINE

By A. Hasselbring

Annual Meeting, Montreal, March 1917.

At the Magpie mine, Michipicotan district, Ontario, is a body of siderite. It is now being mined and made into a merchantable ore by roasting. This roasting, which is done in cement kilns, drives off the volatile and at the same time reduces the sulphur to a point suitable for the blast furnace.

The siderite, together with the other carbonates, occurs as a band standing nearly vertical and striking north east and south west. The band is broken by folding and faults at several points. The width of the siderite that is being mined ranges from 28 feet to 62 feet; the average width is about 42 feet. The carbonate deposit, as a whole, is a sedimentary bed lying between a series of acid and basic flows and tuffs of volcanic origin. The wall rock on the south is a talcose schist with well defined schistosity; while on the north it is an ellipsoidal basalt showing very little schistosity. The contacts are not well defined and are not clean, so that much care is necessary in mining to make sure that no ore is left on the wall and that no rock is broken into the stopes. The schist on the south wall underground has very much the appearance of the ore, but the drill cuttings from the holes give a good indication of when the wall has been reached. The body being mined has an approximate length of 1350 feet. The carbonate band is very much longer than this, but narrows down on either end so that it is not found profitable to mine the ore except in this area.

The siderite, as a whole, is the usual light-coloured ore having a slightly pink tinge due to the presence of manganese carbonate, rhodochrosite; but on either side of the diabase dike, cutting the body, the siderite is dense black ore very much resembling fine grained magnetite. In the white siderite, the volatile runs about 32%, but this volatile gradually decreases near the dyke until it is as low as 12%. The carbonate here contains magnetite in considerable amount and the iron contents of the ore is higher than in the light-coloured ore. The black

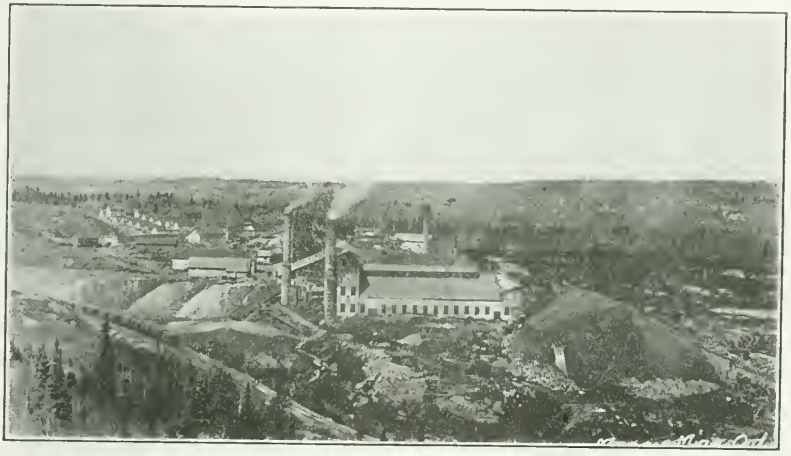
ore is exceptionally hard, so hard in fact that a $3\frac{1}{4}$" piston drill will drill only from 5 to 6 feet of hole per shift. The character of the ore changes gradually as the distance from the dyke increases, so that at about 100 feet from the dyke the siderite is all white.

In roasting the siderite at the Magpie there is a loss in volatile of about 30% by weight, so that nearly three tons of ore must be mined to produce two tons of finished material. Taking this into consideration, together with the fact that the actual roasting operation costs are considerable, it was necessary to devise a very cheap mining system in order to make the operation

The headframe and inclined conveyor to bins at head of kilns at the Magpie mine

as a whole commercially successful. Several mining methods were studied and approximate costs worked out; but before any method was definitely chosen, it was decided to sink the shaft and open up drifts on two main haulage levels to definitely determine the nature of the ground and the material to be mined. The shaft was accordingly started on the north side of the ore body, about 60 feet from the north contact of the ore. The shaft is 24 feet by 8 feet in the rough, and is timbered with 12 inch by 12 inch sets, so that the inside dimensions are 22 feet by 6 feet. It is divided into four compartments, two skip compartments for balanced Kimberly skips, one cage compartment

and one ladder and pipe way. The shaft was sunk 205 feet to the second level, it having been decided to cut 80-ft. levels and to leave a 45-ft. floor pillar on surface. A crosscut was run on each level from the shaft to the south contact of the ore and drifts started thence in either direction. These drifts follow the south contact as nearly as possible. The nature of the ore passed through in the drifts was closely observed and samples taken and analysis made for each 10-ft. section of the drifts. No timber was necessary in any of the drifts, but it was noted that the ore showed a great number of slips or cleavage planes. These



The Magpie roasting plant. In the distance, to the left, part of the village may be seen.

slips have no general direction, but intersect each other at all angles and are extremely smooth. In scaling a new drift, large wedge-shape pieces will fall out from the first blow of the scaling bar; but when a drift is once thoroughly scaled very little material loosens from later blasting. On account of this feature of the ore, it was necessary to decide on a method of mining that would keep the miners always close to the back and under cover. It was therefore decided to use the sub-level stoping system in mining this deposit.

The ore-body was blocked off into three stopes longitudinally, divided opposite the shaft by a 50-ft. shaft pillar and 400

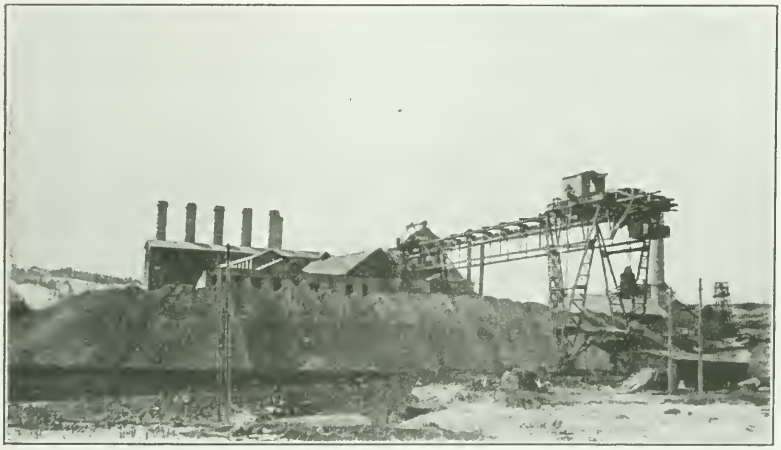
feet west of the shaft by a diabase dike, 100 feet wide, which cuts the body at right angles. This gives three stopes on each level, approximately 400 feet long. To develop these stopes, a raise is made at each end of the block and sub-levels run to connect the raises. The first sub-level is 18 feet above the level. The other sub-levels are 23 feet from floor to floor. On the upper levels three sub-levels are used between levels, but below the second level four sub-levels are used, making the distance between levels 103 feet. After the stopes have been developed in this manner, the raise at the end of the block nearest the shaft is

The Magpie roasting plant, with a view of the ore-bridge and stock yard.

made into a permanent ladder and pipeway. Air lines are carried along the floor of the sub-levels to the far end and mining commenced. Machines are employed in breaking down around the raise at the far end of the block and this opening is enlarged until the stope is completely cut off. The first sub-level is then drawn back about 50 to 60 feet. By keeping the first sub-level back this distance, the muck does not run into the face. This also gives the men working on this sub-level a chance to sand blast a proportion of the larger pieces which break from the upper benches. Most of these drop so that they can be reached

from the first sub-level, but those dropping in the open stope have to be blasted as they come down into the chutes.

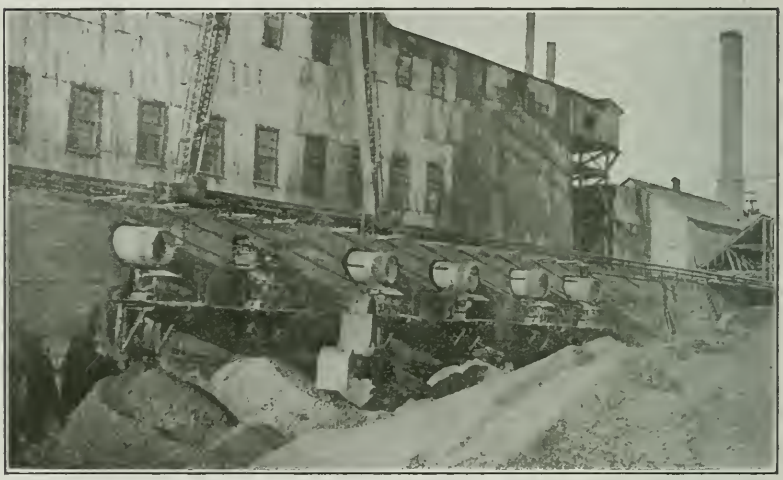
After the stope has been cut off from wall to wall, section cutting is done on each sub-level. At first it was the intention to carry the sub-levels step fashion with the upper sub-levels overhanging the lower ones, but the ground was found to be so full of cleavage planes that these overhanging benches fell when blasting out the section cut, so that now all the sub-levels, except the bottom ones, are carried back together, the face of the stope being vertical. In section cutting the stope, the machine



Magpie roasting plant. A nearer view of the ore-bridge is presented. The stock yard, it will be noted, is well supplied with ore.

is set up in the sub-level and an 8-ft. bench blasted off. This requires five holes, two in front and three behind. These holes are about 7 feet deep and break to the bottom. Very little mucking is necessary for the next set up and there is little scaling, as the back is only 8 feet high. This section cut is carried from wall to wall and the stope holes are drilled in the bench below from the same set ups. The back holes are drilled with stopers after the section cut has been completed. The whole face of the stope is then blasted off with a battery shot. Very little powder is required, either in the section cut or in the stope blast, as there is always an

open face to break to. When a stope on one level has been drawn back to the starting raise, the chutes are taken out, rails and pipe lines are removed and the main level is used as a sub-level. In this way all mucking is avoided, the ore remaining in the bottom of one level which will not run out of the chutes being dropped to the level below. On the bottom sub-level no back holes are used, except in the corners of the stope, as this sub-level is carried higher than the others, thus leaving a thinner block of ground between it and the second sub. The rail and pipe lines, removed from the level which is drawn back, are used



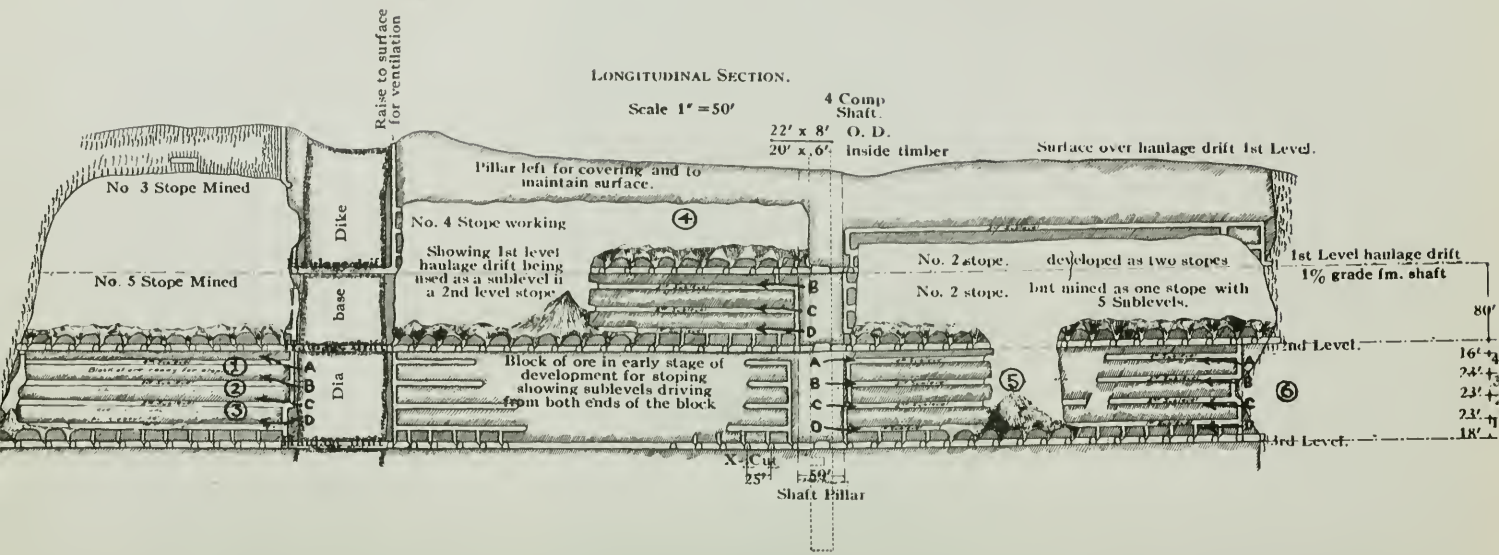
The photograph shows the automatic scales under the cooler discharge.

on the lower level in the development work, so that very few new pipes or rails are required.

On the main haulage level, crosscuts are driven off the main drift at 25 feet intervals. Raises are made from these crosscuts so that the raises are spaced about 25 feet centre to centre each way. The raises extend only to the first sub-level. Ordinary round timber chutes, with three-inch round birch stoppers, are in use in the raises. Much blasting is necessary in the chutes at times on account of the benches falling in large pieces, but otherwise no trouble is experienced in loading cars. All tram-

ming is done by hand, 2-ton cars being used on a grade of 1% in favour of the loads. Cross-switches for spotting cars at the shaft have been discarded. In their stead a truck, running on rails in a shallow pit transversely across the station and about 12 feet back from the shaft, is employed. Cars can be pushed on to this track from any track and spotted for either the skip track or the cage track as may be required. All outbound loaded cars come up the main crosscut on the one track. The mackinaw, if not in use, is spotted for this track. The lead for No. 1 skip lines with this main line. Cars to dump in No. 1 skip come up the main, cross the mackinaw onto this lead and dump directly in the skip. Returning they are backed onto the mackinaw which is then spotted for the return track, and then shoved down the return track through a spring switch out onto the main line and back in again for loading. This spring switch is the only switch on the level. Ordinarily, trammers dump their own cars; but when for any reason it is necessary to speed up the hoisting, dumping is done by a gang of dumpers, (two men), at the shaft. Trammers coming out leave their cars on the main line and go back with an empty from the return track. The gang at the shaft handle cars on the mackinaw, dump them and shove them down the return track. Working in this way 400 to 450 skips can easily be sent up in a shift.

The ore is hoisted with two balanced Kimberly skips, which have a capacity of two tons each and dump directly into the crusher. The hoist consists of a 6-ft. drum, coned at each end and geared to a 150 h.p. wound rotor, three phase, induction motor. This motor is remotely controlled and automatically protected against overloading. It is only, of course, when hoisting from the bottom level that the cone on the drum is of any use, but the motor has no difficulty in starting a loaded skip from any of the intermediate levels, even though no chair is used and the full load is hanging on the rope at the start. The full load speed of the motor gives a rope speed of 750 feet per minute. The signal to hoist the skip is given to the hoistman by a bell which can be rung from only one level—namely, the one from which the most tramming is being done at that time. A skip-tender is stationed there and the other levels ring to him when they want the skip, or when they have finished dumping



①, ②, ③, Block of ore ready for stoping with vertical slice taken out to level above and 1st sub drawn back about 50'.

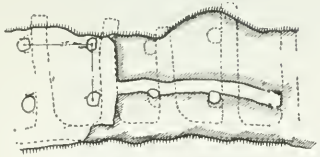
④ No. 1 Stope mined. Showing broken ore left to be recovered in stoping lower levels

⑥ No. 6 Stope working in development.

A 4th Sublevel.
B 3rd Sublevel.
C 2nd Sublevel.
D 1st Sublevel.

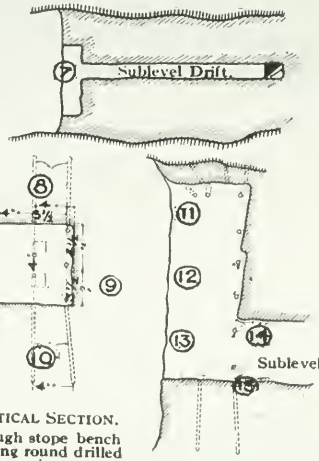
⑥ Showing block of ore being stoped before development is finished b driving sublevels from both ends of the block this is made po sible resulting in reserve ground available when increased pr duction is required and also in doubling speed of recovery almost no increase in final cost.

HORIZONTAL SECTION.
through 1st Sublevel Scale 1" = 20'



Broken line shows haulage drift and crosscuts on level 18' below

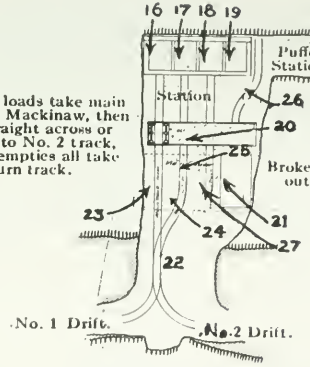
HORIZONTAL SECTION.
through an upper Sublevel
Scale 1" = 20'



VERTICAL SECTION.
through stope bench showing round drilled for section cut.

HORIZONTAL SECTION
through part of an upper Sublevel.
Small figures indicate order in which holes are drilled.

Outbound loads take main track on to Mackinaw, then either straight across or transfer to No. 2 track, inbound empties all take return track.



Rock broken in shaft work is dumped from sinking bucket into 1 ton cars and sent to surface in the cage.

Broken line shows sump cut out under Main X cut.

HORIZONTAL SECTION.
through a Station and Main Crosscut. Scale 1" = 10'

MAGPIE MINE

Sketches to show method of mining

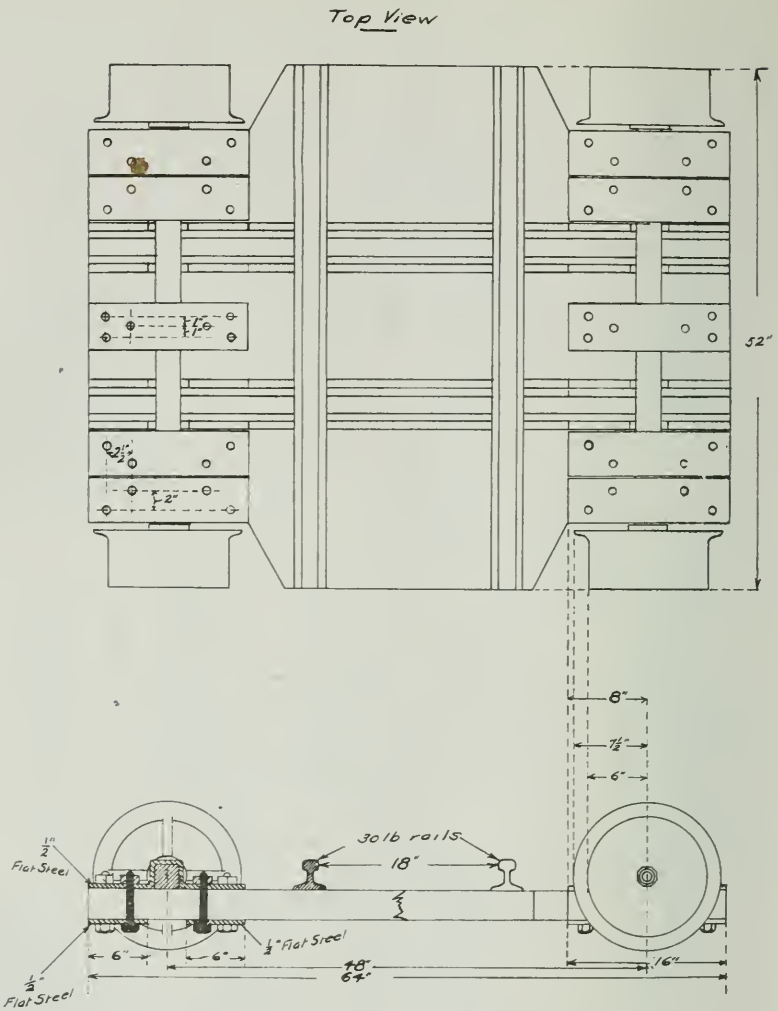
Legend

Ore — Wall rock — Diabase

Siderite unsuitable for treatment

⑦ Stope bench partly cut out.
⑧ Manway back holes along end of bench only.
⑨ Sections to show arrangement of drill holes and order of drilling in stope work scale 1" = 5'
⑩ Stope holes 8'-9'.
⑪ From this set up stope holes in the final corner and end of bench are drilled, also any holes required to get a clean break to the walls.
⑫ Bottom hole 6, 7 and 8 upper holes. 2nd set up. Holes 1-12 are out side holes of section cut.
⑬ 13 and 14 outside holes of section cut. From 1st set up 2 section cuts and 4 stope holes and drilled.
⑭ 6 bottom hole. Holes 7 and 8 are two upper holes of inside row in section cut.

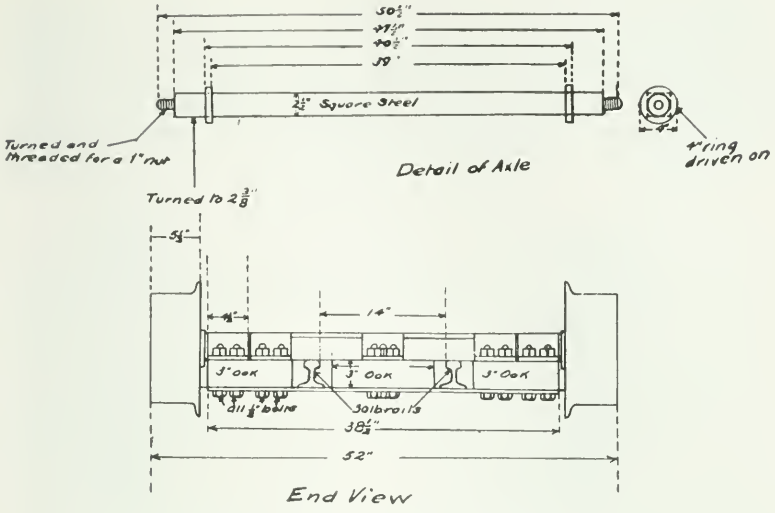
⑮ 10 bottom hole, 11 and 12 upper holes.
⑯ Skip. Comp. No. 1.
⑰ Skip. Comp. No. 2.
⑱ Cage Comp.
⑲ Ladder and Pipe Comp.
⑳ Mackinaw or transfer truck. Mackinaw pit.
㉑ Pump house.
㉒ Spring switch normal position-favoring loads out.
㉓ Main track loads.
㉔ Return track empties.
㉕ Main crosscut.
㉖ From shaft. For handling muck.
㉗ Storage and repair track.



Side view and section on centre line

Sketch of mackinaw at the Magpie mine.

their car; he then relayes the signal to the hoistman. The crusherman feeding the No. 8 crusher, also has a switch by which he can ring the hoistman in case of trouble in connection with the crusher and when he wants to stop the skip before it dumps. This switch also gives the same signal to the skip-tender, so that



Sketch of Mackinaw at the Magpie Mine.

he knows that the skip has been stopped at the crusher. This stopping for a minute or two is fairly frequent, as a big chunk of ore often has to be broken with a hammer before it will go into the crusher. The skips dump into a No. 8 gyratory crusher, which breaks the ore to about 6-inch ring. The black ore from near the diabase dike is, as stated previously, exceptionally hard and it was found that the cast-iron spider, which is practically always supplied with these machines, was not strong enough to withstand the strain of breaking it and had to be replaced by a cast-steel one. Below the No. 8 crusher, the ore passes over a set of grizzly bars and then to two No. 5 gyratory crushers. These are set to about 3 inch, and from these the ore is carried on a 24-inch conveyor belt to the storage bins in the roast plant.

The roasting kilns are 8 feet by 125 feet long and lined with 9-inch hard fire brick. The fuel used is powdered slack coal which gives a temperature of about 1100° c, for about 20 feet in the kiln. This is not hot enough to make the ore sticky and is sufficient to drive off the CO_2 and nearly eliminate the sulphur.

After passing through the roasting kilns, both the light- and dark-coloured ores have the same appearance and are not distinguishable in any way. The finished ore is nearly black in

colour, and comes out of the kilns in a very porous condition in rounded lumps about 2″ in diameter, the large pieces breaking up when passing through the kiln. This finished product has the following composition and is admirably suited for the blast furnace both on account of its physical condition and its chemical composition:

Iron	50.00%
Phosphor	.013
Silica	9.60
Manganese	2.75
Alumina	1.24
Lime	7.69
Magnesia	7.75
Sulphur	.196
Loss on Ign	.000

THE PRESENT POSITION AND FUTURE OF THE IRON AND STEEL INDUSTRIES IN CANADA*

By Corbett F. Whitton

Annual Meeting, Montreal, 1917

In endeavouring to analyze the present situation of the iron and steel industries of Canada in relation to its natural mineral resources, the requirements of its home markets, and the possibilities of future expansion of these markets in Canada and abroad, it is proposed to deal with some of the facts and conditions relating to the commercial and manufacturing side of this subject, including:

1. A description of the principal Canadian iron and steel plants; an account of the sources of their ore and fuels, the cost of assembling these materials at their plants, and the facilities for transporting the products to Canadian and export markets.

2. A statement of the amount and variety of iron and steel products which are manufactured by Canadian plants, as compared with what is imported from the United States and other countries.

3. A statement of the costs of production of various iron and steel products in Canada, as compared with those in the United States. A tabulated statement of the costs of assembling raw materials and transporting finished products from Canadian and American iron and steel centres to various Canadian markets.

4. A consideration of the possibilities of increasing the variety and tonnage of steel products, manufactured in Canada, and suggestions as to possible means of fostering and protecting Canadian steel industries with a view to Canadian trade, and to Imperial and foreign export trade.

---

*Note.—The conditions and data presented in this paper, which was written in 1916, are such as applied to the period 1910 to 1915, and do not represent the present position of the Canadian steel industries, which position has undergone great changes due to the exigencies of the war.

CANADIAN PLANTS AND THEIR PRODUCTS

The principal plants making pig iron and a variety of steel products are:—

Dominion Iron & Steel Company	Sydney.
Nova Scotia Steel & Coal Company	Sydney Mines and New Glasgow.
Steel Company of Canada, Limited	Hamilton, Toronto, Montreal and other points.
Algoma Steel Company	Sault Ste. Marie.

The principal facts relative to the plant, raw materials, and products of these companies are tabulated below.

Dominion Iron & Steel Company

IRON FURNACES AND SUPPLIES	ANNUAL CAPACITY
6 blast furnaces of 250 G.T. capacity each per day	550,000 G.T. basic and foundry pig iron.
Company owns Wabana Ore Mines, Newfoundland	850,000 tons
By-product coke plant, consisting 620 ovens	650,000 N.T. coke
By-product recovery plant for Benzol, Toluol, Naphtha, Naphthaline, and Sulphate of Ammonia	
Plant for manufacture of cement from blast furnace slag	
Limestone quarries in Nova Scotia and Newfoundland	630,000 tons.
STEEL WORKS	**ANNUAL CAPACITY**
10 Basic open hearth furnaces (50-ton).. 3 Basic Bessemer converters (15-ton)..	*400,000 N.T. Ingots; used also for the primary stage of making steel by the duplex process.
35″ 2 high reversing blooming mill	375,000 tons blooms, billets and slabs
16″ Morgan. cont. billet mill	90,000 tons wire rods
Morgan cont. rod mill	60,000 bars
Morgan semi-cont. rod and bar mill with 12″ roughing and 9½″ finishing trains.	75,000 tons bars
16″ Merchant mill	300,000 tons
28″ Rail mill	50,000 tons
Wire department	30,000 tons
Wire Nails	

Nova Scotia Steel & Coal Company

IRON FURNACES	ANNUAL CAPACITY
1 Blast furnace at Sydney Mines of 250 G.T. daily capacity	90,000 G.T. basic and foundry pig iron
1 New blast furnace under construction.	80,000 G.T.
By-product coke plant, consisting of 150 ovens	110,000 N.T. of coke.
STEEL WORKS	**ANNUAL CAPACITY**
5 Basic open hearth furnaces at Sydney Mines	150,000 N.T. steel and fluid compressed ingots
28″ Cogging mill	
20″ Plate mill	Total 60,000 tons of finished, rolled and forged products, including merchant bars, angle bars, tie plates, etc.
18″ and 9″ Rolling mills	
9″ Guide mill	
Hammer and forging presses	
2 Railway spike machines	7,500 tons
MINES	
Ore mines at Wabana, Newfoundland.	600,000 tons
Coal mines at Sydney Mines	800,000 tons
Limestone quarries in Nova Scotia	70,000 tons

The Eastern Car Company, a subsidiary concern, manufactures steel and wooden cars, car wheels, etc. It utilizes a considerable portion of the plates and bars made by the Nova Scotia Steel Co., which has important and growing facilities for steel shipbuilding, and also operates a number of ocean vessels, which carry its ore and coal to export markets.

The plants of the Dominion Steel Company and the Nova Scotia Steel Co. are most favourably situated for the utilization of the enormous iron and coal resources in that territory. A large tonnage of Wabana ore is exported by these companies to Europe and the United States, comprising since 1910, about one-third of the total ore mined, the balance being used in their own blast furnaces producing pig iron suitable for conversion into basic steel, although containing rather high percentages of phosphorus, silicon, and sulphur. The coal supply also is

very well suited for coking and steam purposes. The cost of transportation to these plants both of ore from Newfoundland and coal from Cape Breton mines is very low, compared with that in other important iron districts.

Both companies export a large proportion of their products, notably, rails, forgings, and wire products to the United States, England, and other countries, and should be able to expand their plants so as to increase the variety and tonnage of iron and steel products for export; while their low cost of production and shipping facilities should enable them to compete for export trade with producers in the United States or Germany.

The Steel Company of Canada

The Company has two blast furnaces at Hamilton, Ont., of 250 and 350 G.T. daily capacity, and an annual capacity of 200,000 tons basic foundry and malleable bessemer iron. The iron ore is purchased mainly from the U.S. Lake Superior mining companies. Connellsville 'beehive' coke is used entirely. A by-product coke oven plant is projected. Limestone is purchased from quarries within a close radius. This plant is the largest one in Canada that is entirely dependent on outside supply sources for raw materials. Ore brought from Lake Superior cannot pass through the Welland Canal at present, but is unloaded at Point Edward on Lake Huron, whence it is carried by rail to Hamilton. This involves extra expense as compared with any other blast furnace plants located on the Great Lakes, but this expense will cease after the deepening of the Welland Canal, which will allow large ore vessels to come to Hamilton.

STEEL WORKS	ANNUAL CAPACITY
(Hamilton Works)	
11 Basic open hearth furnaces	400,000 N.T. ingots
34″ Reversing blooming mill	350,000 tons blooms and billets
18″ Morgan cont. billet mill	
Morgan cont. combination rod and bar mill	100 000 wire rods
14″ and 10″ Bar mills	40,000 merchant bars
3 Railway spike machines	10,000 tons spikes
6 Mill sheet plants	25,000 tons sheets

STEEL WORKS	ANNUAL CAPACITY
(Ontario Works)	
20″ Bar mill 10″ Bar mill 9″ Guide mill Car and locomotive axle department Heavy shape forge department	100,000 tons of merchant bars, angle bars, tie plates, etc., including a certain proportion of iron bars.
(Canada Works, Hamilton)	
Wire department	25,000 tons
Wire nail department	240,000 kegs
Other finishing departments are:—	
Tacks, wood screws, machine screws, bolts, nuts and rivets	3,500 tons
(Belleville Works)	
18″ Muck mill	30,000 tons iron and steel bars, merchant and horse shoe bars.
12″ Bar mill	50,000 kegs
9″ Bar mill	20,000 kegs pressed, ship, drift and railway spikes.
Horse shoe department	
Spike department	
(Montreal Works)	
18″ Muck mill 18″ Bar mills (2) 12″ Merchant mill 12″ and 9″ Belgian mills (2)	Products, bar iron and steel, horse shoe bars, angle bars, nail and washer plate. 60,000 tons
Other finished products are:—	
Horse shoe department	110,000 kegs
Wire department	30,000 tons
Wire nail department	275,000 kegs
Horse nail department	30,000 boxes
Cut nail department	30,000 kegs
Bolt and nut department	130,000 kegs bolts, nuts, rivets, washers, etc.
Spike department	55,000 kegs pressed and railway spikes.
Pipe mills, ¼″ to 4″ butt-weld pipe, black and galvanized, couplings and nipples.	25,000 tons

STEEL WORKS	ANNUAL CAPACITY
(*Dominion Works*)	
Wire department	25,000 tons
Wire nail department	125,000 kegs
Other products are:—	
Woven wire fence, gates, barbed wire, bale ties, wire cable	10,000 tons
(*Brantford Works*)	
Bolt and nut department	10,000 kegs of bolts, nuts and rivets
(*Swansea Works*)	
Bolt and nut department	90,000 kegs of bolts and nuts 10,000 kegs of rivets
(*Gananoque Works*)	
Forge department	Carriage and carriage top hardware, auto and miscellaneous vehicle forgings
(*London Works*)	
Wire and nail department	36,000 kegs wire nails

In order to manufacture such a variety of products, a considerable tonnage of rolled iron and steel in the form of bars, wire rods, nut flats, horseshoe bars, plate, etc., which are included in the total output, are used for further conversion in finishing departments and are not sold in the shape of rolled products. While among Canadian industrial firms the Steel Company of Canada is only the third largest producer of steel ingots, it produces the greatest quantity and variety of manufactured steel products, such as foundry pig iron, bars, horseshoes and nails, pipe, spikes, wire of all kinds, wire nails, tacks, rivets, bolts, nuts, washers, wood screws and wire goods. Many of these articles are manufactured in Canada, solely by this company, and are imported only in small quantities. The company operates plants in eight different cities from Fort William to Montreal, and has exceptional facilities for supplying at a low delivery cost the principal Canadian markets.

Algoma Steel Corporation

BLAST FURNACES, ETC.	ANNUAL CAPACITY
3 Blast furnaces:—	
2 of 250 G.T. daily capacity	350,000 G.T. Bessemer and basic pig iron
1 of 450 G.T. daily capacity	
By-product coke plant, consisting of 110 ovens	432,000 N.T.
By-product recovery plant and sulphuric acid plant	

The company owns the Helen and Magpie mines situated about 11 miles north of Michipicoten Harbour. The Helen mine produces hematite ore, averaging about 55% iron. The Magpie mine produces siderite ore which after roasting yields about 50% of iron. When market and mining conditions warrant, a certain tonnage of ore from these mines is sold to other furnaces in Canada and in the United States. The Company also owns several undeveloped iron ore deposits; extensive coal properties in West Virginia, producing 6,000 tons per day; and limestone properties in the neighbourhood of Sault Ste. Marie.

STEEL WORKS	ANNUAL CAPACITY
10 Basic open hearth furnaces	400,000 N.T.
2 (5-ton) Bessemer converters	270,000 tons, formerly used for duplexing, now dismantled.
1 (20-ton) Bessemer converter	180,000 N.T.
1 (200-ton) Basic tilting furnace, for duplexing	
35″ Reversing blooming mill	
Rail mill	350,000 tons
18″ and 12″ Merchant mills producing light rails and structural shapes, splice bars, tie plates and merchant bars	105,000 tons
Bolt and spike department for railway spikes, bolts and nuts	

The Algoma Steel Corporation is very well equipped for producing a large tonnage of rails and rail fastenings, as well as bars and light structural shapes. The favourable situa-

tion of the works in respect of ore supplies and of water shipping facilities has enabled the Company to market a considerable tonnage of rails in the United States in competition with the largest American producers. The position near the head of the Great Lakes affords access to markets of the Northwest and Pacific coast, as well as to Lake Ontario and Eastern points on the St. Lawrence.

The tonnage of light structural shapes rolled is only a small proportion of the total Canadian consumption, but the company could enlarge its plant for the manufacture of heavier structural shapes and plates for which there is a large market in Ontario and Western Canada.

The Canadian Furnace Company

The Company, which produces foundry pig iron for the Canadian foundry trade, has one blast furnace of 350 G.T. daily capacity, and an annual capacity of 125,000 tons of foundry, malleable and basic pig iron.

The Iron Industry in The United States

In comparison with the Canadian plants, the physical and economic position of the principal iron and steel producers in the United States which export an enormous tonnage into Canada, may be considered briefly. The iron industry in the United States leads the world in respect of its natural resources, shipping facilities, and design and efficiency of iron and steel making plants. The U.S. Steel Corporation, which is the largest producer of iron and steel, has played an important part in the development of the export steel trade of the United States, and there are a large number of independent concerns which successfully compete with that Corporation in both domestic and export markets. The Corporation and mostly all of the other iron and steel producers control enormous quantities of ore and coal, which they mine and transport to their plants at a cost considerably below that which must be paid by smaller concerns which purchase their supplies.

We fortunately have very complete and reliable information as to the cost, covering a period of years to 1910, of ore, coke, pig iron, and the various steel products of the United States Steel Corporation; except that allowance must be made for the improvement in methods of manufacture and equipment during recent years, other conditions have not materially changed in the interim; hence present day costs are no doubt considerably lower than ten years ago. The capacity of blast furnaces, open hearth furnaces, and all types of rolling mills of recent design are from 50% to 100% greater than of plants built ten to fifteen years ago. This expansion and development of modern plants in the United States has been made possible by the rapidly growing consumption of their products at home and abroad. As is clearly indicated by the growing volume of imports of iron and steel in the last five years, Canadian industries have not been able to keep pace with the expansion of Canada. Records show moreover that Canadian plants have not increased the variety of their products to any great extent even in recent years, and this is principally due to the fact that they had some protection for the products which they had been making for years; but with regard to new and unprotected products, they were dubious of being able to meet the competition of the United States in Canadian markets. That this fear has been well grounded can be proved by a comparison of the production of Canadian mills for a few years past with their productive capacity. In the circumstances it is evident that there has been no possibility of Canadian producers engaging in export trade, except at rare times when high prices have enabled the steel plants located on the Atlantic seaboard, whose cost of production is considerably lower than at works elsewhere in Canada, to find a market for a portion of their products in foreign countries.

Canadian Productions and Requirements of Iron and Steel

The foregoing figures enable comparisons to be made between the total production by Canadian plants of various classes of iron and steel products, and the total consumption in Canada, represented by the sum total of the production and the imports, less the exports.

Pig Iron

The following table shows the amount of the production, imports, and consumption of pig iron in Canada, and compares the production with the consumption and with the producing capacity of the Canadian plants.

Total Pig Iron Used and Produced in Canada, Gross Tons

	Production	Imports	Less Exports	Total Used	Prod. % of Used	Capacity	Prod. % of Capacity
1911	824,368	186,300	.....	1,010,668	70	.....	....
1912	912,878	244,000	.....	1,156,878	77.7	1,240,000	74
1913	1,015,118	211,500	.....	1,226,618	74.8	1,340,000	78.2
1914	705,972	70,300	12,300	764,000	92.3	1,460,000	48.5
1915	825,420	42,350	15,450	852,320	96.7	1,460,000	56.5
1916	1,043,978	51,900	20,800	1,075,078	97.2	1,460,000	71.5

It will be noted that the maximum production in the year 1913 was only 78.2% of the total capacity of Canadian blast furnaces, while in the year 1914, the production amounted to only 48.5% of the total capacity. In other years the range is between these two limits. The result is that, as the operation of furnaces is limited to less than 70% of their capacity, it is practically impossible to reduce the cost of production of pig iron to a low and uniform basis; for as will readily be understood furnaces operating continuously at their normal capacity are able to show minimum costs, while, under reduced operations, costs will increase nearly in the inverse ratio of the percentage of capacity at which they are operating.

Although these conditions apply to the total production of pig iron of all kinds including, basic, bessemer, foundry, and other classes of iron, it should be pointed out, that practically all Canadian steel works produce sufficient pig iron for the manufacture of open hearth and bessemer steel made in their plants, and the imported pig iron is therefore principally of the foundry grade.

A comparison of the production, imports, and consumption of foundry pig iron in Canada affords a still more striking

illustration of the position of Canadian furnaces. The figures are presented in the following table:

Foundry Pig Iron, Gross Tons

	Production	Imports	Total Used (Export Incl.)	Production % of Used
1911	190,324	186,300	376,624	43.5
1912	194,208	244,000	438,208	44.4
1913	225,231	211,500	436,731	51.7
1914	174,346	70,300	244,646	71.3
1915	125,769	42,350	168,119	75.0
1916	164,501	51,900	216,401	76.1

The percentage of Canadian foundry pig iron produced in relation to the total domestic consumption has ranged in the last six years from 43% to 76%. The latter represents the conditions in 1915, when the total consumption of the country had fallen to a very low limit and the Canadian producers were therefore forced to market their product even at prices prohibiting a profit, or else face the alternative of suspending furnace operations. The normal capacity of furnaces making foundry pig iron is not known, but it may be assumed safely that there are only two furnaces, (one having been in operation only since the year 1913) of which the total annual capacity is about 245,000 tons a year. On this basis the maximum rate of production, attained in 1913, was only 92% of their capacity, but as in years of maximum production, prices are generally profitable enough for steel makers' blast furnaces to be used for making foundry iron for sale, the production was probably not more than 80% of the capacity thus enlarged.

The duty on pig iron is of considerable importance both to makers of foundry pig iron and of steel products made by the straight Bessemer or duplex processes, as one ton of iron is used per ton of steel produced; while in the straight open-hearth process from 40% to 70% of pig iron is used in combination with scrap. Bessemer steel plants have been mainly converted into duplex plants, whereby basic iron is used, being first partly purified in the Bessemer converter, and the final stage of conversion into steel performed in the open hearth furnace. About one ton of iron is used by this method per ton of steel produced. The process is more rapid than the straight open hearth one, but

can only be carried out economically when cost of pig iron is relatively low and scrap high.

Ontario.—The six coke furnaces now in operation have capacities as follows:—

	Number of Furnaces	Daily Capacity	Annual Capacity
Algoma Steel Co.	3	250, 250 and 450	350,000 G.T.
Canadian Furnace Co.	1	350	125,000 G.T.
Steel Co. of Canada	2	250 and 350	200,000 G.T.
Total annual capacity			675,000 G.T.

In addition, there are the following furnaces which have not been in blast or operated continuously for some time:

	Number of Furnaces	Daily Capacity	Annual Capacity
Canada Iron Foundries, Ltd.	*2	125 and 250	125,000 G.T.
Atikokan Iron Co., Ltd.	1	125	45,000 G.T.
Standard Iron Co.	1 charcoal	65 tons	

	Production of Pig Iron—All Grades	% Installed Capacity
1912	527,000 G.T.	75
1913	586,500 G.T.	80
1914	497,500 G.T.	61
1915	450,200 G T.	55
1916	624,300 G.T.	92.5

This tonnage was divided as follows among the different grades:

	Basic and Bessemer	Foundry and Malleable
1912	322,800 G.T.	194,200
1913	361,300 G.T.	225,200
1914	323,200 G.T.	174,300
1915	324,400 G.T.	125,800
1916	442,600 G.T.	181,700

Nova Scotia.—There are now seven furnaces in operation, having capacities as follows:—

	Number of Furnaces	Daily Capacity	Annual Capacity
Dominion Iron & Steel Corp.	6	250 G.T. each	550,000 G.T.
Nova Scotia Steel & Coal Co.	1	250 G.T.	90,000 G.T.
Now building	1	225 G.T.	80,000 G.T.
Total annual capacity			640,000 G.T.

*The largest furnace has been bought and moved to the Algoma Steel Company plant.

Production of Pig Iron by Grades:

	Basic and Bessemer	% Installed Capacity
1912	380,000 G.T.	70
1913	428,600 G.T.	71
1914	203,000 G.T.	32
1915	375,200 G.T.	58
1916	419,700 G.T.	66

Practically no foundry iron has been made in Nova Scotia, except a small tonnage at irregular intervals. Exports of pig iron have amounted to about 4,000 tons in 1914; 17,300 tons in 1915; and 20,800 tons in 1916, mostly from Nova Scotia

STEEL PLANTS

Most of the important plants producing steel ingots and castings by the bessemer, open hearth, and duplex systems have been described before, and the chief point of interest in connection with the present discussion is the growth of the open hearth and electric steel producing capacity of the country during the last two years.

Production of Steel Ingots and Castings, Gross Tons

	Open Hearth	Bessemer	Other Kinds	Total
1912	645,062	207,569	400	853,031
1913	768,663	273,391	449	1,042,503
1914	556,910	186,158	284	743,352
1915	884,736	22,521	5,498	912,755
1916	1,245,488	10,968	30,053	1,286,509

In 1916, Nova Scotia produced 492,500 tons or 38.4% of the total; Quebec, 72,250 tons, or 5.6% of the total; and Ontario, 716,000 tons, or 56.0% of the total.

Rails and fastenings.—The following table gives the returns of imports and production, the capacity of Canadian mills, and the proportion of Canadian production to consumption.

Rails and Fastenings Produced and Used in Canada, Gross Tons

	Production Rails	Production Fastenings	Imports Rails and Fastenings	Total Used and Exported	Production % of Used	Production Rails % of Cap.
1911	360,547 (Est.)	37,000	86,750	484,297	82	68
1912	423,885	52,157	139,800	615,842	77.2	80
1913	506,709	54,839	163,000	724,548	77.5	95.5
1914	382,344	34,165	37,600	454,109	91.5	72.2
1915	209,752	9,406	10,850	230,008	95.2	41.3
1916	81,497	6,479	12,500	100,476	87.7	13.5

Total annual capacity: Rails 530,000 tons; in 1917, 650,000 tons.

It will be seen in comparison with pig iron and other steel products, that Canadian rail mills up to 1914 have been able to operate at a comparatively high rate of production. The principal factors responsible for this condition have been the rapid expansion of Canadian railroads; a comparatively high duty, namely, $7.00 per net ton; and the favourable shipping facilities of the mills by land and water with regard to the points of destination. The tonnage of steel rails exported is not reported nor known accurately; but on several occasions, Canadian rails in quantity have been marketed in the United States at a figure below the American price, in some cases in spite of an import duty, but in recent years, with free entry. Exports in 1915 were about 58,500 G.T. Inasmuch as the price of rails in the United States has been maintained at a fixed level of $28.00 per G.T. for a number of years, it is not surprising that Canadian mills, with the advantage of $7.85 per G.T. duty, have been able to sell the greater part of their production at home; and even in recent years, since the American import duty has been removed, to market a considerable tonnage of rails in the United States.

Iron and Steel Bars.—Under this heading will be included a large variety of products rolled on mills ranging in size from 20″ down to 8″, such as narrow plates, nail plate, tack plate, washer plate, and merchant bars up to 12″ in width and 2″ in thickness, and all sizes of bars up to about 6″ in diameter. This classification also includes small structural shapes rolled on merchant mills such as angles up to 4″ and small channels and beams up to 6″. The latter are produced by one mill, only, namely that of the Algoma Steel Corporation.

The figures for production, imports, and consumption are as follows:—

Production of Merchant Mill Products

(Total Annual Capacity 608,000 net tons)

	Production	Imports	Total Used	Production % of Used	Production % of Cap.
1911	344,760	.....	.....	....	59.5
1912	373,257	208,000	578,257	64.6	64.4
1913	392,340	201,500	593,840	66.2	65.3
1914	218,125	65,100	283,225	77.1	36.4
1915	328,737	101,500	430,237	76.4	54
1916	707,823*	111,400	819,223	86.4	....

*The production includes all shell steel rounds rolled on rail mills and merchant mills.

The imports shown above include only the following items from the Customs Reports: "Bar iron or steel, rolled, etc., exclusive of wire rods of all kinds, shafting, and rolled material of greater value than 3½c. per lb." Rolled iron and steel, hoop, scroll, or strip, 13 gauge and thicker, which is exclusive of hoop iron 12 and 13 gauge and thinner, etc." There is also included in the tonnage of imports: "Rolled iron and steel angles, beams, channels, and other shapes not further manufactured than rolled, 35 lb. per lineal yard and less."

The situation of the rolling mills in Canada is particularly disadvantageous. In all about sixteen concerns operating rolling mills. In 1913 there were twenty works in operation, namely: four in Nova Scotia, one in New Brunswick, four in Quebec, nine in Ontario, one in Manitoba, and two in Alberta.

These mills (seventeen 20″, 18″, and 16″ mills, and twenty-three 14″, 12″, 10″, 9″ and 8″ mills) are capable of rolling all the ordinary sizes and shapes of merchant, agricultural, and small structural bars, but are not equipped, as far as is known, to roll the following products: Bands under ½″ and 16 Ge. including all kinds of hoop, scroll, and strip steel, planished and otherwise special in finish; beams and channels under 6″, except one mill owned by the Algoma Steel Corporation; tees, and small special shapes under 6″; corrugated, ribbed, and other patented reinforcing bars; miscellaneous sections such as special agricultural beams and plates, nut, tire, tractor and rim sections, U bars, and other special deformed and miscellaneous shapes.

The greater part of these special shapes are rolled in the United States on what are called 'specialty mills' especially designed and equipped for producing large tonnages, their costs being accordingly much lower than in the case of mills equipped for producing a greater variety of sections. Hence United States operators have made special efforts to obtain all the tonnage offered both in Canada and the United States in order that these mills might be operated at the highest rate of production possible. There is no doubt that Canadian consumers have been able to obtain the best products and deliveries from these mills, and Canadian mills have therefore been forced to accept the balance of the tonnage obtainable—mainly standard sections and sizes

costing more to produce as more frequent changes are necessary—this business having been divided up among many concerns. The widely distributed markets of Canada, involving long hauls from the principal manufacturing centres to the points of consumption, have encouraged the establishment at various points of many small rolling mill plants, whose maximum annual capacity does not as a rule exceed 25,000 tons. These mills have depended for their existence on securing, and have made every effort to secure, the greater part of the local trade in standard shapes and qualities. The competition of these small plants has severely cut into the business of the larger plants which are equipped with full complements of standard merchant mills and with steel plants for furnishing their own requirements of steel. The result of this competition has been the reduction of prices to a minimum, but without any compensating increase in the variety of products.

Hoops and Bands.—The returns in respect of imports of rolled bands, hoops, and strip steel 14 guage and thinner are as follows:

Year Ending March	Imports—N.T.
1913	18,006
1914	14,662
1915	10,242
1916	13,712
1917	16,813

Steel hoops are used chiefly in the manufacture of barrels, buckets, boxes, etc., and range in size from $\frac{3}{8}''$ to 3″ wide, and $\frac{1}{40}''$ to $\frac{1}{10}''$ thick. Light bands are used for wooden tanks, vehicles, cars, etc., the size ranging from $\frac{3}{8}''$ to 6″ wide and from $\frac{1}{8}''$ to $\frac{3}{16}''$ thick. These are rolled from small billets usually on a 10″ or 8″ semi-continuous or continuous mill. The variety of light hoops and bands used in Canada would cover practically a full range of these sizes, to roll which would require at least four mills ranging from 14″ to 8″, with an annual capacity of about 150,000 tons. This is the usual number and size of mills installed in plants in the United States where hoops and bands are rolled, and it would be out of the question to install and operate mills of this kind in Canada for the small tonnage required.

Structural Shapes.—There are no records of Canadian production of light structural shapes of 6″ and under, rolled on

bar mills; although at least four mills ranging in size from 10″ to 18″ roll angles 4″ and under, small channels, beams for agricultural implements and vehicles. These mills also roll angle bars and tie plates, of which the production has been given under the heading of rails; but so far as is known, the Algoma Steel Corporation and Nova Scotia Steel & Coal Company are the only concerns that roll shapes such as small channels, beams, etc., and the variety and tonnage of their output is not known.

Imports of these have amounted to a large tonnage in the past few years, being as follows:

Light Structural Shapes

Year Ending March	Imports—N.T.
1913	89,459
1914	97,582
1915	27,122
1916	39,455
1917	45,908

In the United States practically all such light structural shapes are rolled on bar mills or light structural mills especially designed for rolling these sections as well as angle bars, tie plates, rail joints and other special sections.

In recent years imports into Canada have been:

Heavy Structural Shapes

Year Ending March	Imports—N.T.
1913	200,678
1914	212,772
1915	64,981
1916	66,571
1917	74,260

In these figures is included a considerable tonnage of heavy shapes weighing over 35 lb. per lineal yard such as I-beams over 20″ and channels over 15″ which require a special heavy structural mill to roll, so that there would hardly remain enough tonnage of lighter shapes of this class for another mill to roll, such mills having an annual capacity of at least 300,000 tons. A light structural mill which would roll angles up to 5″ or 6″ and light beams and channels under 8″, also angle bars, tie plates, and heavy implement shapes, would have an annual capacity of about 180,000 tons. Moreover, a large range of sizes and shapes which would have to be rolled in order to supply all the requirements of the country, would be beyond the capacity of any standard structural mill.

There is no doubt that the past two years have witnessed a very severe depression in steel construction in Canada, and the prospects are good for a considerable increase in structural shapes which will be used in this country. At the present time, the duty on light structural shapes is the same as on rolled bars namely, $7.00, but on heavy shapes it is $3.00 per N.T. The question of whether the latter duty would afford sufficient protection to any Canadian concern willing to risk the heavy cost of installation of structural mills, will be dealt with later on under the heading of "Cost of Production."

Plates.—There is one plate mill in Canada. This is operated by the Nova Scotia Steel & Coal Co.; but this being a 20″ mill, is only capable of rolling light plates up to about 30″ in width. The large tonnage of plates used in Canada under ½″ in thickness would probably range from 30″ up to 120″, used chiefly for bridge construction, car, tank, engine and boiler work. These are imported into Canada under four classifications, as below noted:

"Universal plates" are imported by manufacturers of bridges, structural work and cars. The returns of imports under this item are:

Year Ending March	Imports—N.T.
1913	52,646
1914	57,763
1915	22,219
1916	25,035
1917	21,504

The next range of sizes is entered for import under the description of "Plates not less than 30″ x ¼″." The imports of this class have been:

Year Ending March	Imports—N.T.
1913	56,411
1914	54,832
1915	22,231
1916	27,356
1917	23,884

Both of these classes are subject to an import duty of $3.00 per ton under the general tariff.

Another class is entered as "boiler plates not less than 30" x ¼" for use exclusively in the manufacture of boilers." These are entered free, and the annual imports have been:

Year Ending March	Imports—N.T.
1913	21,535
1914	20,592
1915	5,233
1916	5,858
1917	8,039

In addition to the foregoing, there are other plates entered as "rolled iron or steel sheets or plates, sheared or unsheared, and skelp sheared or rolled in grooves." These are sheets or plates thicker than 14 guage of a class made in Canada and are subject to a duty of $7.00 under general tariff. The imports of this class have been:

Year Ending March	Imports—N.T.
1913	42,120
1914	40,384
1915	14,335
1916	17,245
1917	23,591

The imports in tons of these four classes of plates have been:

Year Ending March	Imports—N.T.
1913	172,712
1914	173,571
1915	64,018
1916	75,494
1917	77,018

The first class of plates mentioned above, which are known as 'universal plates,' range in size from 12" up to about 48" in width, and are rolled on a universal plate mill, which, in addition to its horizontal rolls, which reduce the material in thickness, has vertical rolls on either side, which regulate the width of the plate. These vertical rolls are adjustable to any width up to the size of the mill. The edges of the plates are regular and straight and do not need to be sheared.

The second class of plates described as "not less than 30" wide x ¼" thick" and boiler plates of the same dimensions, are

usually the product of a sheared plate mill, in which the steel is reduced to the desired thickness, but the edges are not rolled or regular. These plates must be sheared when cold to the desired width and length or into special shapes as ordered.

The fourth class of plates mentioned above are usually rolled on a universal plate mill and vary in width from 12″ up to 40″.

Skelp is also rolled on this kind of mill and is imported as raw material for the manufacture of punched and sheared hardware, metal parts, etc., such as hinges, washers, and other similar articles.

It is apparent that the maximum consumption of universal plates in Canada has not exceeded 90,000 to 95,000 tons annually, and of sheared plates not over 75,000 to 78,000 tons, these quantities covering a very large range of widths and sizes.

A universal plate mill rolling universal plates up to 48″ wide and sheared plates up to 72″ wide would have an annual capacity of about 150,000 tons. Larger sizes would require a sheared plate mill. The smaller universal plates under 30″ could be rolled on a small universal mill together with skelp. The annual output of such a mill would be about 100,000 tons.

There would appear to be a favourable opportunity for the installation of two such mills in Canada, as the prospects of an increase in the total consumption of plates of various sizes appear assured. The question of adequate protection would, however, have to be considered. It would be more favourable to the manufacturer of plates in Canada to establish a uniform duty on plates of all kinds.

Sheets.—Sheets of any kind are not made at present in Canada, although a plant was built at one time for the purpose of manufacturing them. The attempt was not successful, but the plant has recently been purchased and removed to Hamilton by the Steel Company of Canada, Limited, which will commence manufacturing sheets of 14 guage and thinner about the beginning of 1918.

The comparative statistics of imports of various classes of sheets are as follows:

"Rolled Iron or Steel Sheets, Polished or Not, 14 Gauge and Thinner"

Year Ending March	Imports—N.T. Dutiable	Free
1913	66,063	7,377
1914	39,125	15,954
1915	29,054	7,260
1916	46,512	1,333
1917	50,784	2,258

"Canada Plates, Russia Iron, Terne Plates and Rolled Sheets of Iron or Steel Coated with Zinc, Spelter or Other Metal, of all Widths and Thickness, N.O.P."

Year Ending March	Imports—N.T. Dutiable	Free
1913	11,974	7,069
1914	8,176	10,192
1915	8,791	5,733
1916	10,134	2,078
1917	12,279	525

"Galvanized Sheets, Flat and Corrugated."

Year Ending March	Imports—N.T. Dutiable	Free
1913	28,111	28,095
1914	16,996	33,758
1915	15,130	20,185
1916	17,441	6,542
1917	10,529	455

There are a few other classifications of imported sheets which do not amount to any significant tonnage and will be omitted from the total tonnage. All of the above sheets are subject to a 5% duty when imported from the United States and are free from the United Kingdom. The totals for all these classes are:

Year Ending March	Total Imports—N.T. Dutiable	Free
1913	106,148	42,541
1914	64,291	69,904
1915	52,971	33,178
1916	74,087	9,953
1917	73,592	3,238

In view of the fact that after the war Imperial tariff walls may be raised by increasing the duties on all products imported from all foreign countries outside of the Empire and the Allied Nations, it is possible that a fair protective duty on sheets may be imposed under the general tariff with a moderate preferential duty.

There is no apparent reason why the whole of the tonnage of sheets now being imported from the United States and England should not be made in Canada. Sheet mills do not cost as much to build and equip as most modern rolling mills such as are required for bars, plates, and structural shapes. The process of rolling sheets differs from the process of rolling these other products, as the rolls, instead of having specially designed grooves which form the shape of product, are perfectly smooth. Black sheets are rolled from sheet bars about 8″ wide and from ½″ to 1″ thick cut in lengths of 30″. These are rolled on a continuous mill from a large billet. After the sheets have had a number of passes between the rolls, they are reduced to a certain thickness after which they are doubled so that two or more thicknesses pass between the rolls at the same time. This doubling is repeated and the sheets come out in a pack so closely united by pressure of the rolls, that it is necessary to pull them apart in separating individual sheets. In reducing the sheets by various passes, they are carried by hand from one stand of rolls to the next. The furnaces for heating the sheet bars and annealing the finished sheets are simple in design and operation. There is no complicated machinery for the handling of the product and the principal item of cost is that of labour. Sheet mills can be established in units of various sizes and capacity according to the market requirements. The tonnage imported would, however, include a wide range of sizes, qualities, and finishes which would require different styles of mills to produce economically. It would therefore appear that if an import duty were imposed for the protection of the Canadian manufacturer that an *ad valorem* duty would be fairer and more advantageous than a specific duty, having regard in particular to the fact that labour costs enter so largely into the costs of manufacture.

Sheets are mainly used for building construction, not only for the walls and roofs of sheet metal buildings, but also in many various small and fabricated forms. Annealed sheets enter into the manufacture of a great variety of Canadian products such as stoves, stamped and drawn work, and utensils and vessels of various kinds. They are also used in large quantities for electrical purposes.

In addition to the black and galvanized sheets used in manufacturing and construction work, there is also a large tonnage of tin plate imported into this country for the manufacture of cans, containers, and other small parts punched from tin plate.

Tin plate is made from black sheets, of a better finish and having a smoother surface suitable for tinning. The additional processes are pickling, annealing and cold rolling. The first process removes impurities from the surface; annealing makes the material pliable; and cold rolling insures perfect smoothness and correctness in gauge. The tin is then applied by drawing the plates between rolls immersed in molten tin, thereby giving a smooth bright coating to the black plates. Roofing plates, as distinguished from galvanized sheets, are usually coated with a mixture of lead and tin and are called terne plates.

The imports of tin plate are shown above under the heading of "Canada plates and rolled sheets coated with zinc, spelter, or other metals." The greater part of tin plate used in Canada was formerly imported from Great Britain, but of recent years, the United States has become the largest exporter to Canada.

Pipe, Tubing, and Fittings.—The greater part of the wrought iron and steel pipe used in Canada under 10″ in diameter is made by three concerns, namely: Canadian Tube & Iron Company, Montreal; Page-Hersey Iron Tube & Lead Company, Toronto; Steel Company of Canada, Limited, Hamilton. The first and last mentioned make only butt-weld pipe 4″ and under in diameter, while the Page-Hersey Company also makes lap-weld pipe ranging from about 4″ to 10″ in diameter.

The total production of wrought pipe on Canada has not been reported, but may be estimated fairly closely from the tonnage of skelp imported by these manufacturers. The imports in recent

years of "wrought pipe not over 4" in diameter," have been as follows:

Year Ending March	Imports—Value
1913	$486,067
1914	375,012
1915	144,309
1916	117,588
1917	368,521

Skelp

Year Ending March	Imports—N.T. Dutiable General	Free—Not over 4¾"
1913	112,996	1,033
1914	101,036	560
1915	94,317	372
1916	95,240	.....
1917	66,244	1,684

Practically all this skelp is made from bessemer steel and is imported from the United States. It is all rolled on special skelp mills consisting of about ten stands of continuous rolls like a continuous sheet bar or billet mill. For the heavier grades of skelp the slabs or sheet bars from which it is rolled are first broken down in a small universal mill. The larger sizes of skelp are rolled entirely on small universal mills. The annual capacity of such mills would run from 75,000 to 175,000 tons. No doubt one mill would be able to furnish the greater part of the sizes used in Canada, namely, for pipe from 2" to 10" in diameter Sizes of skelp for pipe smaller than 2" can be rolled on bar mills, but there is none of the continuous type, rolling small flats, bands, or skelp, in Canada.

The greater tonnage of tubing which is imported into Canada at present is large lap-weld pipe over 4", of which a great quantity is used for gas, oil, and mining purposes.

The total value of imported "wrought pipe over 4"" has been:

Year Ending March	Total	Imports—Value Over 4" up to 10"	Over10"
1913	$1,586,452	.....	.....
1914	698,186	.....	.....
1915	384,122	190,201	193,921
1916	238,288	85,882	152,406
1917	373,803	312,297	61,506

Since 1915 separate returns have been made for pipe over 10″ in diameter.

The duty on the former class is 30% general tariff and on the latter 15% general tariff, due to the lower cost of production per ton of large sizes of pipe. The imports of this class of material have decreased so rapidly that there is little room for doubt that Canadian mills will gradually furnish the whole of the tonnage needed in Canada, where there are not as many grades and thicknesses required as in the United States.

The above classes cover what is known as standard, extra heavy, double extra heavy, oil country, line and drive pipe used extensively in mining and development work in the United States.

Boiler Tubes.—These are not manufactured at all in Canada, but have been imported in considerable quantities from England, and the United States. They are made from extra refined iron and very soft steel, and are generally lap-welded, although a considerable portion is seamless. They are imported free of duty.

The value of imports for the last five years has been:

Year Ending March	Imports—Value
1913	$903,016
1914	886,699
1915	356,992
1916	364,143
1917	1,119,222

The class of pipe, second in point of quantity imported into Canada is: "Iron or steel tubing, plain, galvanized, riveted, corrugated or otherwise specially manufactured."

The value of the imports of this class of pipe for the last four years has been:

Year Ending March	Imports—Value
1913	$1,014,005
1914	1,325,069
1915	395,026
1916	287,716
1917	228,094

In this class is included all kinds of fabricated pipe such as culvert, drain, locked joint, spiral, riveted, chimney pipe, etc.

The rate of duty is 30% under the general and 20% under the preferential tariff, the greatest proportion being imported from the United States. Pipe of this class is never made by steel plants or rolling mills but is the product of smaller manufacturers making a wide variety of fabricated goods from sheet steel and plates. This also applies to "iron tubing brass covered, imported by manufacturers of iron and brass bedsteads," of which the imports also represent a considerable sum annually.

The establishment of skelp mills by one or other of the steel plants in Canada will no doubt lead to an increase in the number of tube mills, producing a greater variety of pipes and tubes than are now made. The duty of 30% to 35% on pipe up to 10" is a fair measure of protection to encourage the establishment of mills equipped to make the other classes of pipe, including boiler tubes required in Canada.

"Fittings of iron and steel" for pipe of all descriptions are imported in enormous quantities. The value of the imports in the last five years has been:

Year Ending March	Imports—Value
1913	$1,266,171
1914	1,020,564
1915	729,472
1916	452,345
1917	718,694

An immense variety of fittings for all classes of pipe are included under this heading, namely, cast iron of all sizes up to 36", small wrought iron, steel and malleable fittings for wrought pipe, also all the miscellaneous joints and fittings required for fabricated pipe. A very large proportion of these fittings would comprise valves of every description, of which there is an immense variety imported into Canada. This branch of business is suited to manufacturers especially equipped to undertake it and not for steel makers.

Wire Rods

These are made in Canada by two concerns, namely, The Dominion Iron & Steel Company, and The Steel Company of Canada, Limited.

The former company installed its mill in 1904, and in the year 1913, built wire and nail departments in order to utilize its

surplus of wire rods. The latter company started to produce rods in 1913, and now furnishes a large tonnage of wire rods to the manufacturers of wire for sale or for making wire nails.

The production and consumption of wire rods for the past four years is given in the following table:

Production of Wire Rods, Gross Ton

	Production	Imports	Total Used (Incl. Exp.)	Production % of Used	Production % of Cap.
1911	76,617	.....	.....	....	85.2
1912	64,082	83,850	147,932	43.3	71.2
1913	68,048	62,800	130,848	52	46.5
1914	59,050	58,900	117,950	50	32.8
1915	114,829	61,900	176,729	65.1	63.9
1916	174,490	52,280	226,770	77.0	97.0

It will be observed that in the year 1913, when the second rod mill was established at Hamilton, the total production of the Dominion did not increase to any great extent, and in 1914, was actually less than during any of the preceding three years. However, the tonnage of wire rods imported was decreased considerably, and, during the year 1915, 65% of the total consumption of the country was produced in Canada, while the percentage of capacity at which Canadian mills were able to operate increased considerably over that of either of the preceding two years.

Of the wire rods imported from the United States, a considerable quantity was used by manufacturers of chain, but a large proportion was also imported by manufacturers of wire and wire nails for use in their own factories, on account of the extremely low prices at which wire rods were obtainable during the past few years. A considerable tonnage was also imported for the manufacture of wire products, which were afterwards exported, especially since the year 1914.

The export trade in wire and wire products has developed enormously since that time especially between Canada and the United Kingdom and other British possessions. The duty of $3.50 per net ton on wire rods was granted from July 1st, 1914, and has already proved a material benefit to the rod makers,

without apparently affecting the manufacturers of wire and wire nails. At the present time practically all the wire nails used are made in Canada, a condition which has existed for some time owing to the duty of 60c. per keg. The following table shows the production and imports of wire nails for the past five years:

Production of Wire Nails in Canada

	Production Kegs 100 lb.	Imports Cut Nails	Imports Wire Nails	Total Used and Exported	Production % of Cap. (1915 capacity 2,000,000)
1912	1,490,000	13,594	47,813	1,551,407	77
1913	1,520,000	3,867	31,141	1,555,008	78
1914	1,144,000	4,689	19,379	1,168,068	58
1915	1,636,000	315	10,617	1,646,932	82
1916	1,757,000	4,600	86,872	1,848,472	92

In addition to wire rods imported by manufacturers of wire for conversion in their own factories, a tonnage of wire almost as large has also been imported direct by consumers as is shown in the following table giving the imports of the more important classes for the past five years.

Imports of Wire

12 Months to March	1913	1914	1915	1916	1917
FREE					
Bessemer and 'Homo spring wire	1,014	925	576	980	1,631
Barbed wire	22,306	11,764	16,270	19,467	22,621
9-12-13 Ge. galv. wire	41,169	35,960	37,912	35,297	18,843
Total free	64,489	48,649	54,758	55,744	43,095
DUTIABLE					
Wire of all kinds, n.o.p. (20%)	5,909	5,228	3,435	2,978	4,111
Woven wire cloth and netting (30%)	1,770	2,498	2,118	2,000 (Est.)	1,170 (Est.)
Wire fencing between 9 and 14 Ge. (15%)	827	699	911	1,000	312
Total dutiable	8,506	8,425	6,464	5,978 (Est.)	5,893 (Est.)
Total dutiable and free—N.T.	72,995	57,074	61,222	61,722	48,988

It will be noted that the bulk of this tonnage enters free in the shape of wire for fencing, and for the manufacture of woven wire fence, also for telephone and telegraph lines, etc.

The tonnage of free wire imported forms 85% to 90% of the total imports. If the total imports of wire are added to the total consumption of wire rods as given in a previous table, it will afford an idea of the approximate total tonnage of wire products of all kinds consumed in Canada. The figures are:

	Total Rods used plus Nails and Wire imported—N.T.	% made from Can. Rods
1912	286,600	22.4
1913	229,900	29.6
1914	212,600	27.8
1915 (Est.)	270,300	41.5
1916 (Est.)	390,900	44.7

It is evident from a consideration of the figures presented in the foregoing tables, so far as the bulk of the products manufactured by the primary or basic iron and steel industries of Canada are concerned, that their present equipment and capacity is a great deal more than sufficient to supply the total demand for these products in Canada; and that important and valuable opportunities are presented Canadian manufacturers to supply a considerable proportion of the products which are now imported wholly from foreign countries, provided that they can obtain the necessary protection and support to engage in this business with a reasonable assurance of profit.

Comparison of Cost of Production of Iron and Steel in Canada and The United States.

The bulk of the export trade of the United States in iron and steel products has been developed from the beginning and is now being carried on extensively and principally by the United States Steel Corporation, the greatest undertaking of the kind in point of size, resources and variety of products in the world, and other large concerns located in the Buffalo, Cleveland, Youngstown, Pittsburgh and Chicago districts.

The United States Steel Corporation, as is well known, owns vast ore deposits in the Lake Superior region with which its plants in these districts are supplied. It also owns and operates the

mining railroads which transport the ore to the upper lake ports, the boat lines which carry ore to the lower lake ports and the railroads which take the ore from the vessel to the blast furnaces. All the other principal independent companies, practically own and control their supply of ore as well as vessels transporting it on the Great Lakes. Those iron and steel industries that are not so fortunately situated in this respect are forced to buy their ore in the open market, where a very large proportion of the total ore consumed is sold each year by mining companies that do not own blast furnaces or are not financially associated with blast furnace interests. The market price of ore is fixed each fall for the succeeding year, and the prices of various grades sold by different mines are based on four standard grades, namely, Old Range, Bessemer and Non-Bessemer; and Mesabi, Bessemer and Non-Bessemer. The metallic contents of these ores is taken as a standard and the price of all other ores is adjusted according to their variation from this standard in metallic contents, phosphorus and moisture.

The published prices for the last seven years of these four grades, which are based on delivery at Lake Erie docks, have been as follows:

	Old Range		Mesabi		Frt. Rate G.T. to Lake Erie
	Bess.	Non-Bess.	Bess.	Non-Bess.	Ports
1910	$5.20	$4.20	$4.75	$4.00	$0.55
1911	4.50	3.70	4.25	3.50	.50
1912	3.75	3.00	3.50	2.85	.40
1913	4.40	3.60	4.15	3.40	.45
1914	3.75	3.00	3.50	2.85	.40
1915	3.75	3.00	3.45	2.80	.40
1916	4.45	3.70	4.20	3.55	.50

The standard iron content for Bessemer ores is 55%; for non-Bessemer, 51.50%.

As all the figures in this paper relative to production, imports, and manufacturing capacities of plants are for the years 1911-1915 inclusive, the average of the ore prices for these years may be taken as representing the normal price during that period. This average is $3.08 per G.T. for Mesabi iron ores unloaded on Lake Erie dock.

The comparative costs of making pig iron and steel in the various processes of manufacture from pig iron to the finished

product, may now be considered, and in this connection the estimated cost of mining and transporting Mesabi ore to Lake Erie ports in the years 1910 and 1915, respectively, is given in the following table:

	1910	1915
Cost of ore at mine, including labour, expense, depreciation and royalty	$1.23	$1.23
Railway freight	.41	.40
Lake freight	.62	.40
Total cost lower lake ports	$2.26	$2.03
Quoted market price	4.00	2.80
Profit	1.74	.77

The freight rates for the year 1910 as shown represent the net transportation costs of the United States Corporation, using its own railroads and lake vessels, whereas in 1915, the lake freight charge of 40c. is the open market contract price. Taking the average of these two costs at lower lake ports as $2.15 and comparing it with average market price of $3.08 for the 5-year period, the average price paid in the open market by companies not controlling their own ore mines has been 93c. above that of the Steel Corporation and other large United States companies. In addition to this difference, it should be pointed out that the average freight rates given above would include a profit of not less than 15c. a ton on ore to companies owning their own mines, railroads, and lake vessels. It seems fair, therefore, to assume that small concerns which buy their ore in the market—and in this category are included all the Ontario iron and steel plants using ores imported from the United States—have to pay over $1.00 per ton on ore more than the large American companies. Assuming that an average of two tons of ore is required to make one ton of pig iron, the difference in increased cost of pig iron for ore alone is therefore at least $2.00.

Most of the large concerns in the United States also own coal mines from which is obtained all the coal required by them for coking, miscellaneous heating, and steam purposes. These coal lands are situated in Ohio and Pennsylvania where, in the Connellsville region mainly, the better grade of coal is found. These fields are in easy reach of Pittsburgh, and the cost of transportation on coal is therefore very small. The greater part

of the coal is brought to the coke ovens at the blast furnaces for coking. Coal used in the Chicago district is brought partly from Pennsylvania and partly from Illinois and Indiana. This coal is high in ash and volatile matter, and must be mixed with the better grade of coal from the Connellsville and Pocohontas region. It has been found that the average selling price of coke for the 5-year period, 1910-1915, was about $2.00 per N.T. at ovens, and the average cost to the U.S. Steel Corporation was about $1.60. For the purpose of estimating the present cost of pig iron, a profit of 25c per ton has been assumed. The total profit on ore and coke per ton of iron would therefore be $2.40. It is apparent, therefore, that the cost of transportation of ore and coal or coke, is a very large factor in the cost of production of pig iron, and the total cost of assembling raw materials is the deciding feature affecting the economic location of blast furnaces.

The freight rates on ore and coal from Lake Superior mines to blast furnaces at three Ontario points and in the three principal American districts are tabulated hereunder. The three Canadian plants located in Ontario are selected for reference because they are principally concerned in the competition originating from the United States, especially on account of drawing their supply of ore and coal from the same districts as do the American furnaces.

Freight Rates and Total Assembling Cost on Ore and Coke

	Freight on Ore (G.T.)			Freight on Coke (N.T.)			
	Lake	Rail and Handling	Per ton Iron	Rail and Boat per ton Iron (Basic)	(Fdry.)	Total Cost per ton Iron (Fdry.)	(Basic)
Sault Ste. Marie	$0.30	$0.10	$0.80	$2.20	$2.64	$3.44	$3.00
Hamilton	.40	.65	2.10	2.40	2.88	4.98	4.50
Port Colborne	.40	.10	1.00	2.40	2.88	3.88	3.40
Buffalo	.40	.10	1.00	1.85	2.22	3.22	2.85
Pittsburgh	.40	.88	2.55	.75	.90	3.45	3.30
Chicago	.40	.10	1.00	2.00	2.40	3.40	3.00

	Fdry.	Basic
Average cost at Sault Ste. Marie, Hamilton and Port Colborne.	$4.10	$3.63
Average cost at Buffalo, Pittsburgh and Chicago	3.36	3.05

Sault Ste. Marie, Port Colborne, Buffalo and Chicago are all lake points and the rail haul to the blast furnaces is therefore

negligible. A charge of 10c. for switching and unloading has been assumed as the approximate cost of handling at these points. The total freight charge on ore per ton of iron is based on the estimated consumption of two tons of ore per ton of iron. The freight on coke, in the case of Sault Ste. Marie, includes the cost of a rail haul to Buffalo and boat to the Sault; while in the case of Chicago, the freight rate is based on an estimate of the proportion of coal brought from Pennsylvania together with that mined in Illinois and Indiana. The freight rate on coke per ton of iron is based on 1.2 tons of coke to one of iron, which equals 2,400 lb. per G.T. of pig iron. This practice is based on the average consumption of coke per ton of iron, in the past four years, reported by the American Iron & Steel Institute.

Comparison of Cost of Basic and Foundry Pig Iron in the United States and Canada.

	United States		Canada		Difference	
	Basic	Foundry	Basic	Foundry	Basic	Foundry
Price per ton ore at lower lake ports	$3.08	$3.08	$3.08	$3.08	....	....
Less lake freight	.40	.40	.40	.40	....	....
Price at upper lake ports	$2.68	$2.68	$2.68	$2.68	....	....
Price of coke at Connellsville ovens	2.00	2.00	2.00	2.00	....	....
Cost of ore per ton of iron	5.36	5.36	5.36	5.36	....	....
Cost of coke per ton of iron	2.00	2.40	2.00	2.40	....	....
Assembling cost of ore and coke	3.05	3.36	3.63	4.10	$0.58	$0.74
Cost of ore and coke per ton of iron	$10.41	$11.12	$10.99	$11.86	$0.58	$0.74
Furnace conversion cost						
Limestone	$0.50	$0.55	$0.55	$0.60	....	....
Labour	.55	.55	.80	.80	....	....
Steam	.35	.35	.70	.70	....	....
Surplus gas credited	(.55)	(.55)	(.85)	(.85)		
Repairs and maintenance	.10	.10	.15	.15	....	....
Supplies and expense	.35	.35	.70	.70	....	....
Relining fund	.18	.18	.25	.25	....	....
Contingent fund	.05	.05	.05	.05	....	....
Total conversion	$1.53	$1.58	$2.35	$2.40	$0.82	$0.82
Total cost of pig iron	$11.94	$12.70	$13.34	$14.26	$1.40	$1.56
Less profit on ore and coke included	2.40	2.50	....	....	2.40	2.50
	$9.54	$10.20	....	....	$3.80	$4.06

Comparison of Cost of Basic and Foundry Pig Iron in the United States and Canada—Continued.

Considering difference in assembling cost, total cost would be:			Credit for coke by-products			
Sault Ste. Marie	....	....	$12.51	$13.36	$1.40	....
Hamilton	....	....	14.21	15.14	1.45	$1.75
Port Colborne	....	....	....	14.04	1.40	1.70
Buffalo	$9.34	$10.06	....	....	1.35	1.65
Pittsburgh	9.79	10.29	....	....	1.25	1.50
Chicago	9.49	10.24	....	....	1.40	1.70

Extra for benzol by-products 70c.

Selling Prices		United States Basic (Valley)	United States Foundry (Buffalo)	Canada Basic	Canada Foundry
Average	1912	$13.90	$14.89		
	1913	14.77	14.87	Average price of all kinds	
	1914	12.80	12.84	of *Pig Iron Imported* into	
	1915	13.78	13.98	Canada 12 months end-	
	1916	19.87	20.63	ing March 31st.	
Minimum	1912	12.26	13.62		
	1913	12.83	13.07	1914	$14.10
	1914	12.50	12.08	1915	15.15
	1915	12.50	12.44	1916	13.80
	1916	17.88	18.00	1917	24.80

Comparative Cost of Open Hearth Ingots, Gross Ton

	United States	Canada	Difference
Cost of basic pig iron per G.T.	$9.54	$13.34	....
Per cent of pig iron used	55%	45%	....
Per cent of scrap used at $11.00 per G.T.	40%	53%	....
Per cent of ore, cinder and scale used at $5.00 per G.T.	5%	2%	....
	100%	100%	....
Cost of materials used			
Pig iron	$5.25	$5.98	....
Scrap	4.40	5.83	....
Ore, cinder and scale	.25	.10	....
	$9.90	$11.91	....
Lbs. of steel ingots produced from 100 lb. charge	89 lb.	89 lb.	....
Cost per G.T. ingots	$11.10	$13.40	$2.30
Reagents (Ferro-alloys, etc.)	.35	.40	....
Limestone and fluxes	.15	.20	....
Fuel	.65[1]	1.60[2]	....
Labour	.65	1.00	....
Moulds and stools	.10	.15	....
Materials in repairs and maintenance	.10	.15	....
Supplies and expense	.50	.85	....
Rebuilding fund	.35	.55	....
Total conversion	$2.85	$4.90	$2.05
Total cost of open hearth ingots per G.T.	$13.95	$18.30	$4.35

[1]Natural gas. [2]Prod. gas.

Estimated Cost of Structural Shapes (Over 35 Lb. per Lin. Yd.)

	United States	Canada	Difference
Cost of ingots per G.T.	$13.95	$18.30	$4.35
Lbs. of ingots per 100 lb. of blooms	115 lb.	115 lb.	....
Lbs. of scrap recovered for 100 lb. of blooms	12 lb.	12 lb.	....
Cost of Blooms			
Cost of waste	$0.78	$1.43	....
Cost of labour	.35	.50	....
Cost of fuel	.15	.50	....
Expenses	.45	1.00	....
	$1.73	$3.43	$1.70
Total cost of blooms	$15.68	$21.73	$6.05
Cost of Rolled Shapes			
Lbs. of blooms used for 100 lb. of shapes	118 lb.	118 lb.	....
Lbs. of waste recovered for 100 lb. of shapes	15 lb.	15 lb.	....
Cost of waste	$1.17	$2.27	....
Cost of labour	1.75	2.50	....
Cost of fuel	.20	.65	....
Expenses	1.65	2.25	....
	$4.77	.$7.67	$2.90
Total cost of structural shapes, per G.T.	$20.45	$29.40	$8.95
Total cost of structural shapes per 100 lb.	.913	1.315	.402

Selling Prices—(Pittsburgh Base)	Average	Minimum	Average Price of Imports from United States	
1912	$1.30	$1.11	Year ending March 1913	$1.33
1913	1.42	1.24	Year ending March 1914	1.41
1914	1.16	1.05	Year ending March 1915	1.28
1915	1.30	1.10	Year ending March 1916	1.50
1916	2.50	1.87	Year ending March 1917	2.76

Estimated Cost of Large Billets, Gross Ton

	United States	Canada	Difference
Price of ingots per G.T.	$13.95	$18.30	$4.35
Lbs. of ingots used per 100 lb. of billets	115 lb.	115 lb.	....
Lbs. of scrap recovered per 100 lb. billets	12 lb.	12 lb.	....
Cost of waste	$ .78	$ 1.43	....
Cost of labour	.50	.70	....
Cost of fuel	.20	.50	....
Expenses	.75	1.25	....
	$2.23	$3.88	$1.65
Total cost of large billets per G.T.	$16.18	$22.18	$6.00

Cost of Small Billets, Gross Ton

(Re-rolled direct from large billets)

Price of large billets	$16.18	$22.18	$6.00
Lbs. of large billets used per 100 lb. of small billets	103 lb.	103 lb.	....
Lbs. of scrap recovered per 100 lb. sm'l billets	3 lb.	3 lb.	....
Cost of waste	$0.16	$0.33	....
Cost of labour	.30	.55	....
Cost of fuel	....	....	....
Expenses	.40	.60	....
	$0.86	$1.48	$0.62
Total cost of small billets per G.T.	$17.04	$23.66	$6.62

Estimated Cost of Merchant Bars, Gross Ton

(Base sizes including small structural shapes)

Average cost of large and small billets G.T.	$16.61	$22.92	$6.31
Lbs. of billets used per 100 lb. of bars	115 lb.	115 lb.	....
Lbs. of scrap recovered per 100 lb. of bars.	12 lb.	12 lb.	....
Cost of waste	$ 1.17	$ 2.12	....
Cost of labour	2.25	4.00	....
Cost of fuel	.35	.60	....
Expenses	1.30	2.75	....
	$5.07	$9.47	$4.40
Total cost of merchant bars per G.T.	$21.68	$32.39	$10.71
Total cost of merchant bars per 100 lb.	.968	1.44	.472

Merchant Bars

	Selling Price (Pittsburgh Base) Average	Minimum
1912	$1.25	$1.10
1913	1.38	1.22
1914	1.15	1.05
1915	1.31	1.10
1916	2.48	1.87

Large Billets (Per G.T.)

1912	22.74	19.40
1913	25.56	20.00
1914	20.08	19.00
1915	23.14	19.00
1916	45.02	35.50

Average Price of Imports of Merchant Bars (Including extras)

	All Countries	United States
Year to March, 1913	$1.45	$1.40
Year to March, 1914	1.55	1.50
Year to March, 1915	1.44	1.40
Year to March, 1916	1.81	1.81
Year to March, 1917	2.65	2.65

Average Price of Imports of Light Structural Sections

Year to March, 1913	1.40	1.40
Year to March, 1914	1.96	1.96
Year to March, 1915	1.34	1.33
Year to March, 1916	1.51	1.51
Year to March, 1917	2.79	2.79

Estimated Cost of Sheared Plates, Gross Ton

	United States	Canada	Difference
Cost of ingots per G.T.	$13.95	$18.30	$4.35
Lbs. of ingots used per 100 lb. of slabs	119 lb.	119 lb.	....
Lbs. of scrap recovered per 100 lb. of slabs at $11.00 per G.T.	15 lb.	15 lb.	....

Cost of Slabs

Cost of waste	$1.01	$1.83	....
Cost of labour	.40	.55	....
Cost of fuel	.10[1]	.50[2]	....
Expenses	.40	1.00	....
	$1.91	$3.88	$1.97
Total cost of slabs	$15.86	$22.18	$6.32

Cost of Plates

Lbs. of slabs used per 100 lb. of sheared plates	131 lb.	131 lb.	....
Lbs. of scrap recovered per 100 lb. of sheared plates at $11.00 per G.T.	29 lb.	29 lb.	....
Cost of waste	$1.73	$3.68	....
Cost of labour	1.90	2.50	....
Cost of fuel	.30[1]	.50[2]	....
Expenses	1.45	2.00	....
	$5.38	$8.68	$3.30
Total cost of sheared plates per G.T.	$21.24	$30.86	$9.62
Total cost of sheared plates per 100 lb.	.948	1.38	.432

Estimated Cost of Universal Plates, Gross Ton

Cost of slabs per G.T.	$15.86	$22.18	$6.32
Lbs. of slabs used per 100 lb. of plates	114 lb.	114 lb.	....
Lbs. of scrap recovered per 100 lb. of plate	12 lb.	12 lb.	....
Cost of waste	$ .90	1.78	....
Cost of labour	1.45	2.00	....
Cost of fuel	.40[1]	.50[2]	....
Expenses	1.35	1.90	....
	$4.10	$6.18	$2.08
Total cost of universal plates per G.T.	$19.96	$28.36	$8.40
Total cost of universal plates per 100 lb.	.892	1.265	.373

Selling Prices—United States (Base)			Average Price of Imports into Canada			
	Average	Minimum	Year to March 31st	Univ. Over 12″	Not Less 30″ x ¼″	Boiler Plates Not Less 30″ x ¼″
1912	$1.30	$1.11	1913	1.32	1.37	1.54
1913	1.41	1.20	1914	1.41	1.47	1.60
1914	1.14	1.05	1915	1.46	1.49	1.40
1915	1.29	1.10	1916	1.79	1.66	1.58
1916	2.82	1.90	1917	3.27	3.30	3.79

[1]Natural Gas.
[2]Prod. Gas.

Estimated Cost of Wire Rods, Gross Ton

	United States	Canada	Difference
Small billets	$17.04	$23.66	$6.62
Waste (105% of billets used)	.53	.85	....
Labour	1.50	1.75	....
Fuel	.35	.65	....
Other expenses	1.50	1.60	....
	$3.88	$4.85	$0.97
Cost of wire rods per G.T.	$20.92	$28.51	$7.59

	Selling Price (Pittsburgh) Average	Minimum	Imports of Rods (Not Over 3/8") Year to March	N.T.	Price per G.T.
1912	$26.15	$24.75	1913	91,919	$26.10
1913	28.39	25.25	1914	69,806	27.10
1914	25.50	24.50	1915	65,587	25.00
1915	28.26	24.80	1916	61,750	30.60
			1917	52,300	56.00

Estimated Cost of Black Sheets, Gross Ton

	United States	Canada	Difference
Cost of ingots per G.T.	$13.95	$18.30	$4.35
Lbs. of ingots used per 100 lb. of sheet bars..	115 lb.	115 lb.	....
Lbs. of scrap recovered per 100 lb. of sheet bars	12 lb.	12 lb.	....
Cost of Sheet Bars			
Cost of waste	$0.78	$1.42	....
Cost of labour	.85	1.30	....
Cost of fuel	.30	.50	....
Expenses	1.35	2.00	....
	$3.28	$5.22	$1.94
Cost of sheet bars per G.T.	$17.23	$23.52	$6.29
Cost of Black Sheets			
Lbs. of sheet bars used per 100 lb. of sheets..	116 lb.	116 lb.	....
Lbs. of scrap recovered per 100 lb. of sheets.	15 lb.	15 lb.	....
Cost of waste	1.12	2.10	....
Cost of labour	12.00	15.00	....
Cost of fuel	.60	.90	....
Expenses	3.90	4.50	....
	$17.62	$22.50	$4.88
Cost of black sheets per G.T.	$34.85	$46.02	$11.17
Cost of black sheets per 100 lbs.	1.555	2.06	5.05

Selling Price—(Pittsburgh Base —28 Ge.)			Average Price of Imports (14 Ge. and thinner) (From United States)	
	Average	Minimum		
1912	$2.00	$1.80	Year ending March 31, 1913	$2.32
1913	2.20	1.85	Year ending March 31, 1914	2.43
1914	1.89	1.80	Year ending March 31, 1915	2.16
1915	1.93	1.74	Year ending March 31, 1916	2.25
1916	3.06	2.65	Year ending March 31, 1917	3.54

Total Freight on Ore, Coke, and Foundry Pig Iron, Gross Ton

FROM	Sault Ste. Marie	Hamilton	Pt. Colborne	Buffalo
Freight on ore and coke to furnace, per ton of iron	$3.44	$4.98	$3.88	$3.22
Total freight to				
Head of Lakes	6.44	9.03	7.93	7.27
Hamilton	5.94	4.98	4.88	4.22
Toronto	5.69	5.83	4.98	4.32
Montreal	6.69	6.98	6.03	5.37

Total Freight on Ore, Coke and Steel Products, Gross Ton

(Bars, Plates, Shapes, etc.)

(Based on ½ ton of pig iron to 1 ton of steel)

FROM	Sault Ste. Marie	Hamilton	Sydney	Pittsburgh	Chicago
Freight on ore and coke to furnace per ton of steel	$1.50	$2.25	$0.25	$1.65	$1.50
Total freight to					
Head of Lakes	4.86	6.17	8.87	6.63	6.48
Walkerville	6.43	5.72	8.00	5.17	5.02
Hamilton	9.56	2.25	7.31	5.88	5.73
Toronto	9.56	4.04	7.09	5.88	5.73
Montreal	6.43	5.83	4.95	6.83	8.56
Export—Montreal & Hal.	5.28 (Mtl.)	5.16 (Hal.)	3.05		
Export—New York	....	5.54	....	4.11	6.20

Total Freight on Ore, Coke and Wire Products, Net Ton

FROM	Hamilton	Sydney	Pittsburgh	Buffalo and Cleveland
Total freight on ore and coke to furnace per ton of steel	$2.25	$0.25	$1.65	$3.22
Total freight to				
Head of Lakes	6.17	8.87	6.63	5.96
London	5.16	7.87	5.88	6.51
Hamilton	2.16	7.31	5.88	6.51
Toronto	4.04	8.08	5.88	7.24
Montreal	6.40	4.95	6.83	8.44
Export—Montreal & Halifax	5.16 (Hal.)	3.05	....	....
Export—New York	5.54	....	4.11	5.68

Freight Rates

FROM	Ore per G.T. rail incl. handling		Coke per N.T. (or Coal per ton Coke)	
	Boat	Rail		
To Sault Ste Marie	$0.30	$0.10	$2.20	
Hamilton	.60	.65	2.40	
Port Colborne	.40	.10	2.40	
Buffalo	.40	.10	1.85	
Pittsburgh	.40	.88	.75	
Chicago	.40	.10	(Penn).2.50	(Ill. & Ind.) 1.20
Sydney	(Est.).20	....	.10	

Pig Iron—(Per G.T.)

FROM	Sault Ste. Marie	Hamilton	Port Colborne	Buffalo
To head of Lakes (boat)	$3.00	$4.05	$4.05	$2.45
Hamilton	2.50	....	1.00	1.32
Toronto	2.25	.85	1.10	1.58
Montreal	3.25	2.00	2.15	2.94

Rail from head of Lakes to Winnipeg, $3.01; Calgary, $9.00; Vancouver, $11.20.

Rolled Steel Products

(Bars, Plates and Shapes, per 100 lb.)

FROM	Sault Ste. Marie	Hamilton	Sydney	Pittsburgh	Chicago
To Head of Lakes (boat	15	17½	38½	22¼	20
Walkerville (rail)	22	15½	35	15.7	..
Hamilton (rail);	18	..	31½	18.9	18.9
Toronto (rail)	18	8	30½	18.9	18.9
Montreal	..	..	..	..	..
Export—Montreal	22	13	(Hal.)12½	..	19
Export—New York	..	14.7	..	11	21

Rail from head of Lakes to Winnipeg, 32c; Calgary, 90c; Vancouver, 65c.

Wire Products

(Wire nails, etc., per 100 lb.)

FROM	Hamilton	Sydney	Pittsburgh	Cleveland
To head of Lakes	17½	38½	22¼	12¼
London	13	34	18.9	14.7
Hamilton	..	31½	18.9	14.7
Toronto	8	30½	18.9	17.9
Montreal	18½	21	23.1	23.1
Export—Montreal	13	(Hal.)12½	..	..
Export—New York	14.7	..	11	11

Discussion

Mr. F. P. Jones:—I have had the privilege of reading Mr. Whitton's paper, and I wish to thank him for the trouble, the thought, and the work he has put into the effort. In point of national importance, the iron and steel industry occupies a pre-eminent position. If Canada is to take her place as a manufacturing nation, an iron and steel industry is a pre-requisite. But if Mr. Whitton's calculations and statements are to be accepted as correct, I fail to see how a successful iron and steel industry can be established in Canada. His calculations have, I think, been made on a wrong basis. First of all he takes the production of pig iron in Canada during 1912, as 913,000 tons or 77.7 per cent of the amount Canada consumed, and in 1913, as 1,015,000 tons or only 74.8% of the Canadian consumption. Here Mr. Whitton has represented the theoretical producing capacity and not the actual. Not only that, we have a number of furnaces in Canada that are not equipped to operate. Take the Dominion Iron & Steel Company: Mr. Whitton places its present capacity at 550,000 tons and its maximum production at 350,000 tons. I contend that in 1913 Canada produced every pound of pig iron that it was possible for her manufacturers to turn out, and instead of 77% the furnaces were operated at practically full capacity. As to the figures Mr. Whitton gives in connection with producing capacity, we have no actual statistics of pig iron in Canada, but there is a shortage instead of a surplus. That is shown in his statistics of the imports. The Steel Company of Canada has a pig iron capacity of 200,000 tons and can use every pound of their own pig iron. As a matter of fact, the Dominion Iron & Steel Company never could use over 355,000 tons; while the Nova Scotia Steel capacity is 90,000 tons. However, to be brief, I think on Mr. Whitton's own figures it is fair to assume that we are not in a position to market pig iron, or even to meet our own present requirements.

Then as to wire rods. Mr. Whitton gives the production in Canada in 1915, of rods as 114,800 tons and stated that we used in Canada 179,000 tons. That statement is absolutely misleading. The Dominion Iron & Steel Company has one rod mill producing 90,000 tons of rods in a year. The Steel Company of Canada

has a rod mill which can produce more rods than Canada has ever consumed. These companies could have produced all the rods that were needed, but they put their steel into other products. In estimating the cost I think Mr. Whitton has been unintentionally unfair. He selects for comparison the Lackawanna plant and the plant of the United Steel Steel Corporation—two of the greatest producing plants in the United States. He compares them with plants in Ontario. These two United States' plants probably can produce more cheaply than any other plants in the world, while the plants in Ontario, have the highest operating costs of any plants in Canada. Ontario, has however, something to offset the high cost; it is nearer the big consuming markets. It would have been fair to compare the American cost with that in Nova Scotia. Had this been done the showing would have been very different. Mr. Whitton goes into details as to labour for making pig iron, but I am unable to follow his figures. He states that scrap is cheaper in the United States than in Canada; I do not think that is correct. Scrap is just as cheap in Canada. Canada is exporting scrap to the United States to-day. Mr. Whitton says that it costs $5.24 a ton more to make rails and billets in Canada than in the United States. Algoma and Sydney have exported rails to the United States and have undersold American producers in their own country. The Canadian companies that did this have either made a tremendous loss or Mr. Whitton's figures are wrong. Canada, I am satisfied, will be an iron and steel producing country, but the way to build up the industry is to make provision for additional production in the one line where we are short, and not to scatter our energy by getting into competition with existing mills.

MR. G. A. IRWIN (Algoma Steel Company):—As regards the steel industry in this country, it may be remarked that while the companies operating in Canada have passed through several lean years, they have during the past two years been remarkably successful, due principally to the shell steel in-industry; and I am confident that the future shows much promise. I have in my hands the latest Government records, and to give an idea of the advancement in this Canadian industry, I would point out that whereas our production of steel in the shape of ingots and castings amounted to less than 20,000 net tons in

1895, the production in 1913 (the last year of normal conditions and for which figures are available) shows 1,169,000 net tons. This on the face of it would indicate marked prosperity; but if we look at the matter from another standpoint we shall find that while Canadian production is rapidly increasing, it is still far short of our own requirements, as is shown by the fact that whereas imports of iron and steel products into Canada in 1895 were less than $10,000,000 in value, imports in 1913 were close to $150,000,000 in value. The bulk of these imports are in the shape of structural steel, plates, sheets and bars. Up to the present Canada has not made plates or sheets in any considerable quantity and has only manufactured structural shapes in the smaller sizes which are protected by a duty of $7.00 per ton. In 1914 the Canadian Government adopted a tariff resolution authorizing the Governor in Council, when satisfied that rolled iron or steel angles, beams, channels, and other rolled sections or shapes were being manufactured in Canada in substantial quantities, to impose upon such shapes up to the weight of 120 lb. per lineal yard, the tariff of $7.00 per ton now effective on lighter shapes namely 35 lb. per lineal yard and under, instead of $3.00 per ton as at present existing. This action on the part of the Government has resulted in Canadian steel makers giving serious consideration to the installation of heavy structural mills and it is very probable that within the next year or two Canadian mills will be producing a very considerable portion of the requirements of this country in respect of structural shapes of all sizes.

The Algoma Steel Corporation, Limited, with its works at Sault Ste. Marie, Ontario, has been increasing its steel capacity considerably of late and it is anticipated that when present improvements shall be completed we will have an output of about 50,000 tons of ingots per month. The mills at the Soo were started primarily for the production of heavy rails, the demand for which was then great in Canada. With our increasing steel production we must manufacture a more diversified product to find a ready market, and serious consideration is being given to other lines which it would be best to undertake. I believe that other steel companies are considering the manufacture of fresh

products, and these developments will all assist in making Canada more self-supporting than in the past.

I have reason to believe that more that one company in Canada is considering the installation of a plate mill, especially in view of the probable increase in shipbuilding which will necessitate the use of a large tonnage of steel plates. In 1913, 3,565,000 net tons of plates and sheets were imported into Canada, and it is reasonable to expect that the consumption in this country will increase rapidly once business conditions shall become normal. I do not desire to discuss politics, but would mention that one reason why Canada has not gone into the manufacture of plates and sheets has been the non-uniform tariff. Boiler plates enter Canada free of duty, and plates for the manufacture of ships obtain a rebate of 99% of the duty, so that a very considerable portion of the plate tonnage now consumed enters this country practically duty free. To operate a plate mill successfully a large tonnage would be necessary, and until such time as Canadian makers can be assured of reasonable protection on all varieties of plates it will be difficult to induce capital to invest. I understand, that if the Canadian Government assists shipbuilding by bonus or otherwise, it may perhaps provide some protection for the manufacturers of the steel plates which will be required. Were this done, it would materially alter the situation and give much greater encouragement to prospective Canadian manufacturers than now exists.

The manufacture of sheets is somewhat intricate. Sheets are produced in many qualities—plain black, galvanized, polished, planished, etc. They require to be carefully prepared and carefully packed, and the successful production of sheets requires highly-skilled labour which would be difficult to obtain and hold in Canada. Sheets at present enter Canada from Great Britain free of duty, with a duty of only 5% from the United States, and we could hardly hope to compete successfully against the large mills in England and the United States on the present basis. A protective tariff would need to be assured before Canadian mills would undertake the production of sheets.

Our Government friends have not given a fair show to the makers of wire and wire rods. For many years wire rod manu-

facturers were assisted by a bounty but this has been withdrawn within recent years and, further, the tariff has been altered to permit of the free entry into Canada of rods for use in the manufacture of galvanized wire. During the past couple of years large quantities of wire rods and wire have entered Canada from the United States; but I feel that with reasonable protection a great bulk of this tonnage would be produced in Canada.

As to the general up-building of the steel trade in this country, I agree with Mr. Jones that there is a bright future. Our friends in the East are to-day making handsome profits and so far as I can see they ought to be in a position to produce steel as cheaply as in any other centre in the world. With the enlargement of their present plant and the fact of their being on tide-water, they should be able not only to secure a fair share of the Canadian market but also to compete successfully for export trade. The Steel Company of Canada is exceptionally well situated from a standpoint of distribution. Their mills are in the most thickly populated section of Canada, where large steel tonnage is consumed, and even if their raw material does cost more than at Sydney or the Soo, they have the advantage in distribution.

As you are well aware, the steel manufacturing plants in Canada have, during the past year, been occupied largely in the manufacture of a high-grade steel for munition purposes and I am pleased to be able to state that Canadian mills have been very successful in manufacturing a product which is second to none. The technical experience gained in producing this special steel will be of great benefit to the operating departments of our mills, and should result in Canada being in a much stronger position after the war than heretofore. The important thing for us is to enlarge our plants along the right lines so as to be able to take care of the requirements of the country to the fullest possible extent once normal business conditions shall return. I feel sure that this question is being given a most careful and searching investigation by all of the leading Canadian companies.

Mr. J. G. Morrow:—I agree heartily with what Mr. Jones and Mr. Irwin have said in regard to co-operation and experience in the iron and steel industry in Canada. There are two or three angles from which we must regard this problem and I think Mr.

Whitton has presented it from one angle and Mr. Jones from another. In the case of pig iron, the Hamilton works of the Steel Company of Canada is in close proximity to a large number of manufacturing companies. Some of these are producers of agricultural implements, and pig iron is one of the largest of their importations. For agricultural purposes, this pig iron comes into Canada, with a drawback duty, and they get all the duty refunded except one per cent. Under these conditions we are in open competition with the cheapest makers of pig iron in the world.

In regard to scrap, we had a condition not normal in Canada. We exported scrap because we had an excess and we had not the furnace capacity to use it. With the inauguration of a number of plants, especially electric, the consumption of this material has been increased, and we are now able to take care of all and possibly more than we can produce. Mr. Whitton, I know, has based his calculations on the reports issued by the Governments of Canada and of the United States through the proper departments, so that I am confident that in the main his figures are correct. It must not be assumed that pessimism is the keynote of Mr. Whitton's paper; neither, I take it, is optimism alone sounded in Mr. Jones' address, but I am sure that if the iron and steel industries of Canada will co-operate on some sort of basis for the consideration of their joint interests, to enable them to produce new products and to distribute them economically, many facts which at present seem unreal will be demonstrated and made clear, and with co-operation proper Governmental measures for greater development in the iron and steel industry in Canada may be forthcoming.

DR. J. B. PORTER:—It must be realized that the importance of the steel industry cannot be exaggerated, and the present occasion, and particularly these two interesting papers that are being discussed, give the Institute an exceptional opportunity for bringing the matter before the public and incidentally before the governing authorities.

MR. J. A. DRESSER:—The paper of Mr. Whitton seems to have a national bearing and, very properly, is being discussed in that light. The aim is to get the Dominion into a self-contained position in respect of its raw materials for the production of iron

and steel. This brings us back to the unfortunate position which we had to consider some three years ago, when it was fully realized that our known supply of iron ore that could be used in its natural state was altogether insufficient to meet our needs in iron and steel. Since that time our consumption has increased enormously, the production only slightly, and consequently the general condition is less favorable than before.

Certain ores, or near ores, exist in large quantities but nearly all have to be beneficiated in one way or another to make them fit for use in the blast furnace. Titanium bearing ores are abundant, especially in Quebec, but their profitable reduction has not yet been accomplished. The 'banded' iron ores which are in immense quantities in Ontario, are high in finely intermingled silicates, and require mechanical concentration. Magnetite containing sulphur above smelting limits is found in large quantities, but it requires roasting to remove the sulphur. Siderite, which is already mined in quantity in the Michipicoten district of Ontario, requires to be calcined and in parts further roasted when there is an excess of sulphur. All of these ores require treatment after mining to fit them for furnace use.

Apparently the best attempt to utilize such ores has been that made by The Algoma Steel Corporation of Sault Ste. Marie in treating the siderite of Michipicoten, where more than 200,000 tons have been produced during the past year. This has been done under conditions of high labour costs, but in a favourable market. In brief, it may be said that the operation has attained a measure of success, but still needs improvement to assure its permanent and most economical use. Important efforts have also been made to utilize the other classes of ore that I have mentioned, particularly those of Moose Mountain, Ontario; but at present these efforts are somewhat in abeyance. On the whole, it certainly appears that the effort to provide a domestic supply of iron ore that is most likely of success lies in the way of utilizing, even at a minimum profit, those ores that we know we have in large quantities. The need and opportunity for investigation along this line will not, I am sure, escape the attention of the Honorary Advisory Council for Industrial Research.

The utilization of the ores we have would inevitably lead to more thorough examination of the deposits we already know, and

to prospecting for others. Such detailed examination accompanying mining operations might also disclose concentrations, even though relatively small, or ore of higher grade such as the now nearly exhausted Helen mine of Michipicoten. Neither the prospector nor the geologist can hope or be expected to find in unworked territory either all or the best that it may ultimately produce. Our immediate hope, if any, seems to be in the ores we have in quantity. The prospects are not rosy but they are the best we have of getting a domestic supply of iron ore in eastern Canada.

MR. T. R. DRUMMOND:—I can only speak from a prospector's point of view. I do not pretend to be an authority in any way, but I would like to say a few words about the iron region of the western part of the district of Sudbury. I can say that a large amount of what is at present considered low-grade iron ore occurs there over a distance of about forty miles, and for a width of from 500 to 1,000 feet. Much is massive magnetite, and still farther along is what may possibly develop as hematite ore. Whether these ores exist in very large quantities or not is another question; but there is a considerable amount of ore containing from 40% to possibly 45% of iron. It is absolutely free from titanium and reasonably free from phosphorus and low in sulphur; three important requisites. There is also apparently a considerable amount of ore that would average over 50% of iron.

MR. BELKNAP:—I represent one of the smallest iron industries in Canada, the Standard Iron Co. Limited, with a blast furnace at Deseronto. We have been making charcoal iron there for the last four or five years with more or less success, but there have been difficulties in assembling the raw material. We have experimented at Deseronto with one or two Canadian ores, one of these being a concentrate from the Moose Mountain mine, but up to the present time without any material success. The proportion of phosphorus and sulphur in this concentrate is so much higher than in Lake Superior ore that it is impossible to meet the specification for low phosphorus iron or even standard bessemer. The Moose Mountain people are experimenting with their ore and have succeeded in reducing the sulphur to a great

extent through sintering, but they have not yet been able to get the phosphorus sufficiently low even to meet the standard Bessemer specification. To conduct experiments of this nature on an adequate scale is beyond the scope of any one concern, and it appears to me to be a matter that the Government should initiate and carry out in a comprehensive manner. They have, I know, made many experiments at Ottawa in the treatment of ores from various localities, especially from localities in Ontario; but in our opinion have never reached any definite or satisfactory conclusions.

Mr. C. F. Whitton:—(*Communication to the Secretary*). In reference to Mr. Jones' remarks I desire to submit the following reply:

Figures and Authorities:—The figures for the production of pig iron and various steel products have been extracted direct from the bulletins issued by the Department of Mines and the American Iron & Steel Institute, all of which are compiled from the data furnished them by the producers of iron and steel. I do not think, therefore, that the figures representing actual production can seriously be questioned. Figures representing imports have been compiled from the Government blue books and are unquestionably accurate as far as the classifications permit.

Capacities of Blast Furnaces:—Mr. Jones doubts the accuracy of my figures showing the relation of the production of pig iron in Canada to the total consumption in the country. The figures submitted in the various tables are compiled from the following sources which are believed to be reliable: Department of Mines, Annual Report specifying the number of furnaces at each plant, their capacity, and the number of days each furnace operated. It is an easy matter to calculate from these data the total capacity and production of each plant, and these figures have been made to agree in the aggregate with the total reported production of the country. The data have also been checked, and other information in regard to open-hearth plants, rolling mills, and other finishing departments have been obtained, from the Iron & Steel Works Directory which is issued by the American Iron & Steel Institute.

Comparison of the total production and consumption of pig iron in Canada:—The writer must admit having erred in the figures showing the ratio of production to capacity. The calculations were based on the present capacity of active blast furnaces, and the increase in capacity due to the building of new furnaces in the year 1913 was overlooked. I have therefore revised the calculations on the basis of the actual capacity of blast furnaces active in the various years, which are shown separately under Ontario and Nova Scotia production. The percentages are now as follows:

1912	74.0%
1913	78.2%
1914	48.5%
1915	56.5%
1916	71.5%

I do not think there should be any doubt as to the producing capacity of these blast furnaces; and taking the number of days which each furnace has been reported in blast during the various years, I find that the total production of each plant and all the plants together agree fairly closely with the tonnages reported by each company annually to the Statistical Bureau and also published in their annual reports. Few furnaces, if any, run continuously throughout a year and produce their normal rated tonnage. On looking over the annual reports, I find that in 1913 at the four large blast furnace plants in Canada, comprising 12 blast furnaces, 7 furnaces were out of blast a total of 317 days or 7% of the whole, although that year was the biggest pig iron year prior to 1916. In 1914, three of these furnaces were idle all year and seven others lost 1,068 days, a total of 2,163 days or 46% of the whole. In 1915, 5 furnaces lost 1,314 days or 30%. In 1916, 4 furnaces lost 785 days or 18% of the possible operating time.

I do not think there is any doubt but that these conditions actually represent true operating conditions in past years in Canada, as the pig iron statistics of the United States show that practically the same conditions existed there in the years 1914 and 1915, when their pig iron production dropped to 23,000,000 tons as against an average production of 30,000,000 for previous years,

and 40,000,000 tons in 1916. A large increase in the rate of production of pig iron in the United States in the last half of 1915, indicated that their blast furnaces had only been operating up to that time at a normal rate but when hard pushed, they could get out an additional 10% of their normal tonnage, this rate having been maintained practically ever since. My contention is, that in Canada we have sufficient blast furnace capacity to take care of our past maximum requirements and our present and immediately future requirements if these furnaces are operated on an economical basis and at full production as is being done in the States. In regard to the individual plants, Mr. Jones does not explain why the Dominion Iron & Steel Company has never produced more than 350,000 tons of pig iron in any year although the company has had six blast furnaces available since 1913. I do not think that he would maintain that they cannot produce more than this tonnage. The Algoma Steel Company has operated its furnaces nearly continuously, and yet its production for the 12 months ending June, 1915, was only 213,000 tons, against 312,000 tons produced in the previous 12 months. These facts are difficult to understand unless we know the actual conditions under which these Companies have been operating.

The Steel Company of Canada's record of production is given in the same table with the other steel companies, and I can only say in explanation of the small tonnage produced as compared with capacity, that the number of days lost by our blast furnaces has been due to various causes such as relinings, explosions, business depressions, and in the past year, shortage of coke, all of which have probably affected other pig iron makers at various times.

I do not agree with Mr. Jones when he states that there is a shortage rather than a surplus of pig iron in Canada, either of steel making or foundry grades. Taking the present steel capacity of the various plants, as nearly as the writer can ascertain, the Algoma Steel Company has an approximate capacity of 50,000 N.T. a month or 600,000, N.T. a year. Inasmuch as their pig iron capacity is only 350,000 G.T., it is evident that it had sufficient pig iron only to operate its open-hearth furnaces on the straight open-hearth plan, but not by the duplex process; or about enough pig iron to operate the 20-ton Bessemer converter

continuously and if desired, to duplex this metal. It is evident therefore that this company had no surplus of iron. As to the Steel Company of Canada, its present capacity is 400,000 N.T. of steel a year for which it would use about 150,000 tons of pig iron, assuming that scrap can be obtained cheap enough and in sufficient quantity to use a 40% pig iron and 60% scrap mix. Our furnaces have been producing pig iron at the rate of 200,000 tons annually as long as conditions and the supply of raw materials permitted, so that we expect to have a surplus of 50,000 tons or more of foundry pig iron for sale. The Dominion Iron & Steel Company's steel capacity is reported as 400,000 N.T. annually, its maximum production being 376,000 tons in 1916. Assuming that it would produce all the steel by the duplex process, it would require approximately the same tonnage of pig iron, which would leave the company a surplus, if operating its six blast furnaces at full capacity, of about 175,000 tons. This should afford the company an opportunity to enlarge the steel plant and find an export market for an increased tonnage of finished products.

Taking the consumption of pig iron for steel making as a whole, we find in the report of the Bureau of Mines for 1915, that the maximum consumption for steel making in 1913 was 913,000 N.T., corresponding to an ingot production of 1,170,000 N.T. The production of ingots in 1916 was 1,428,000 N.T. and the pig iron used, 949,000 N.T., leaving 249,000 N.T. surplus. The total steel capacity of the various large plants is now close to 1,500,000 tons and the pig iron used is practically 900,000 G.T.

Assuming the total capacity of blast furnaces at present active and able to operate as 1,400,000 tons, which I think is a fair estimate, there should be a surplus pig iron capacity above requirements for steel-making of nearly half a million tons. The fact that Canada imported a considerable tonnage of foundry pig iron in 1912 and 1913 does not prove that there was a shortage of pig iron in those years, because there was then a pig iron producing capacity almost equal to what exists now, and if market conditions had permitted, the Canadian producers would have been able to fill all the Canadian requirements of foundry pig iron. The present severe shortage of foundry pig iron, both

in Canada and the United States, is due not only to an increase in demand for this grade of pig iron but to the great shortage of raw materials and the excessive demand for steel making pig iron. When this condition is removed there is no doubt that Canadian producers can take care of all the Canadian demand for foundry pig iron.

Wire Rods.—There can be no doubt as to the accuracy of the figures of wire rod production and consumption in Canada as shown in the table. Since the year 1913, in which the Steel Company of Canada's rod mill was built, it has never been able to run the mill double turn on account of low market conditions in the States and the absence of protection. In the middle of 1914, however, a duty of $3.50 a N.T. was granted by the government, except on wire rods for the manufacture of wire fencing, which are admitted free. Owing to the scarcity of steel, we could have operated our rod mill double turn, as the Dominion Iron and Steel Company has done since 1915 (owing to the large export market thrown open to them) except for the fact that our steel capacity was needed for other purposes, principally munitions. When the demand for munitions ceases, we expect to run our rod mill double turn and supply as large a proportion of the home market requirements as we can under the present tariff and market conditions.

Mr. Jones' statement that the Dominion Iron & Steel Company can produce 90,000 tons of rods a year is correct; in 1916 it produced 112,000 tons of which about half was exported. The Steel Company's rod mill, however, cannot produce more than 110,000 N.T. a year, which is a long way short of Canada's total consumption.

Costs.—The writer must admit that he has considered the subject of Canadian iron and steel industries principally from the point of view of the western industries, where we are continually in competition with the chief producers of iron and steel in the United States. All the cost figures have been based on conditions which obtain in Ontario, as the writer was able to obtain reliable data in regard to the cost of ore, coke, transportation and conversion costs for this district but was unable to do more than

guess in regard to the cost of producing iron and steel products at the Atlantic seaboard. I hope the figures are understood to cover only such grounds as I have mentioned, and I trust that those who are able to present similar data in regard to the Nova Scotia industries will do so, as the information is needed for an intelligent consideration of the whole question.

Mr. Jones, probably, has not had time to read or study the figures submitted in my paper showing the transportation costs on finished products from the various steel plants, including Sydney, to the principal market points in Canada; or the combined figures showing the cost of assembling raw materials at the various plants, added to the cost of transportation of their products to various districts.

I would like to draw attention to the economic situation of the several plants in regard to the various market districts.

To head of lakes, whence goods are shipped to the Northwest, Sault Ste. Marie shows the lowest total freight costs on ore, coal, and steel products of $4.86 a gross ton as against the next lowest cost at Hamilton of $6.17. The lowest American cost is at Chicago $6.48.

For Hamilton and Toronto, Hamilton is the cheapest assembling point.

For Montreal, Sydney is the most favourable point, showing a cost of $4.95 against Hamilton $5.83, and Sault Ste. Marie $6.43; the lowest American cost to this point being that of Pittsburgh $6.83.

For export, I figure that Sydney's total assembling costs and freight to Halifax is $3.05 as against Pittsburgh to New York $4.11. There are other American steel companies on the Atlantic seaboard that bring their ores from foreign countries such as Cuba, South America, Spain, and Sweden, but I cannot hazard a guess as to their cost of assembling materials.

Comparing Hamilton and Sydney with Pittsburgh and Cleveland in regard to wire products, it will be seen that Hamilton's total assembling costs and freight on products is the cheapest to London, Hamilton and Toronto, and Cleveland to the head of lakes, but Sydney has the lowest to Montreal and for export.

I heartily agree with Mr. Jones in his recommendation that iron and steel industries in Canada should co-operate on a closer basis for the purpose of discussing and considering their joint interests in regard to extensions of plants, the manufacture of new lines, distributing tonnage to the best economical effect, and avoiding useless competition. Also they could use their associated efforts in presenting to the Government a clearer and more comprehensive view as to the actual protective measures needed for the development of Canadian iron and steel industries.

I do not think it is unfair to compare our situation here in the central part of Canada with the other large industries in the States similarly situated, with regard to supplies of ore and coal, and also their product markets. When you attempt to consider a broad question such as an analysis of the iron and steel industry in Canada, one does not gain anything by closing one's eyes to the fact that the large steel companies of the United States are able to produce their ore and coal and their semi-finished and finished products at much lower costs than Canadian producers. In order to have the intelligent grasp of the whole situation, one must take all these factors into account, and I think that we are better off in appreciating the exact conditions confronting us rather than in trying to hide them. Either we are going to develop the iron and steel industry in Canada, or these larger American concerns are going to increase their strength here.

No one who has any knowledge of the iron and steel industry in Canada can doubt that the Nova Scotia plants have probably as cheap facilities for assembling raw materials as any plants in the world, and should be able to compete successfully in export and Eastern American markets with either Pittsburgh or the southern States' plants. On account of their situation on the seaboard, they are also able to compete successfully on the Pacific seaboard with these and any other producers in the world, and there is no doubt that they have been able to place rails in the United States at a profitable price at times when conditions in Canada permitted them to export rails. Of course both Algoma and Sydney have always had the advantage of a very fair protection in Canada and have been aided in selling rails in the United States, owing to the fact that for years there has existed a rail selling pool, which has fixed the price at $28.00.

Of course American rail producers have been earning large profits on rails, as they probably do not cost the largest producers more than $17.00 a ton, the principal reason being that the railroads and government were content to allow the steel producers a fairly large profit in order that the quality of rails might not suffer and that they might be afforded a sufficient margin in years when railroads were taking a large tonnage to tide them over long periods of shutdowns and depressions. I therefore do not think that my figures are far wrong or can be upset on the grounds that Canadian producers have been able to market their rails in the States. I think this is sufficient proof for the satisfaction of anyone who understands the conditions mentioned above. I wish that Mr. Jones, who has had a wide experience and reputation in the steel business, had considered and presented some figures in regard to the costs of assembling materials and of making pig iron and steel, after which there would not have been much doubt as to the correctness of the figures of costs of other finished products.

OPPORTUNITIES FOR THE ESTABLISHMENT OF AN IRON AND STEEL INDUSTRY IN BRITISH COLUMBIA

By Robert R. Hedley

Western Branch, Vancouver Meeting, March, 1917

It may be observed that we have at present no iron and steel industry in British Columbia, as it is confined to small undertakings which depend wholly on scrap for their source of supply. It is said that we are not yet ready for such an industry; that our market is too limited, that our coal is not sufficiently developed at the Coast and is of inferior quality and too costly to make a suitable metallurgical coke at reasonable price; that the conditions prevailing in our labour market are such as to offer a serious handicap to economic work; and, finally, that we have no hematites as yet available to mix with the prevailing magnetites of the coast. Further it is believed that electric power is not obtainable at such cost as to render electric smelting for pig iron a commercial success at normal prices.

All this is rank pessimism and if persisted in will leave the Province a laggard in the march of progress.

Market.—Without going into figures on this subject, let us admit that the market for products of iron and steel (especially if limited to such as may be produced in a plant of modest rolling capacity) is small; nevertheless it is increasing, and surely it may be extended greatly in the time of great industrial activity that is sure to follow the war. Let us suppose that a beginning be made with an economic unit for blast furnace production of pig, open hearth furnaces for steel, and a rolling mill capable of producing the common sizes of bar, light rail, etc., is it not highly probable, nay, sure, that a profitable market will readily absorb any surplus if offered in the shape of steel billets.

Coal.—Opinions to the contrary notwithstanding, my belief is firm that certain coal from seams of the Vancouver Island collieries, properly selected or washed, and coked in by-product ovens, will produce a superior metallurgical coke. As to cost

there is a lack of data on which to estimate closely. The Canadian collieries are today coking their small coal in bee-hive ovens and supplying the Granby smelter at Anyox with about 3,000 tons per month at an approximate cost of $6.00 per ton. The management states that this absorbs all their small coal, and a further supply only be made by using lump, at a cost of $7.00 per ton. By-product coke may be produced at a substantially lower cost than beehive coke, using coal at the same price. Further, coal can undoubtedly be mined more cheaply on the Island. Estimates have been made that by-product coke can be produced at a cost of $4.50 per ton at the ovens—and this without taking into account the value of the gases and other by-products. Surely this question is worthy of closer investigation than it has hitherto received.

Labour.—Undoubtedly our labour conditions are adverse; but rightly handled, it is conceivable that this handicap may be overcome. Labour efficiency may be augmented by judicious management.

Iron Ore.—It is true that the ores of iron at the Coast are chiefly magnetites, that high grade hematites occur only remote from transportation, and that limonites are a somewhat doubtful asset. Magnetites are often of very superior quality, high in iron and very low in sulphur and phosphorus content. Other deposits however are contaminated by sulphur and copper, but the latter is not an unmixed evil, as it enables one to procure the desirable hematite, or its equivalent, at a low cost. At many points, notably at Birdsboro and at Lancaster, in Pennsylvania, and also in Norway and Sweden, low grade magnetites are crushed and magnetically separated, the resultant product being prepared for the blast furnace either by sintering or briquetting. The advocates of either process claim for their product a superiority over hematite for blast furnace work. Mr. B. G. Klugh,[1] of Birdsboro, Pa., referring to sinter says: "At all times the sinter has shown itself a superior and beneficial material for the blast furnace charge." Mr. Fred. A. Jordan, when manager of the Moose Mountain iron mine, Ontario, said of his briquette which carried over 60% iron: "It has 30% voids and is made up of at least 90% hematite."

[1]Trans. A. I. M. E. Vol. XLIII, page 364.

H. V. Hansell presented a very interesting paper on this subject to the Canadian Mining Institute.[2] The plant at Sellwood has not been in operation for the past two years, for the reason, that it was economically, though not metallurgically, a failure. The ore so treated carried but 36% iron, and the tailing about 7%.

Three years ago Mr. J. B. Tyrrell met at the Treadwell mine a Russian engineer who was having tested by the Gröndal Co. of New York, a Russian ore, which carried 45% iron and 0.75% copper—and he anticipated highly successful treatment.

In this case the separation of ten tons would probably result in six tons magnetic and four tons non-magnetic. The latter by concentration would probably produce one ton of copper ore, making three tons waste. At Moose Mt., at least two tons were required to produce one ton of high grade briquette. Here, so far as my observation goes, one may start with a feed that will carry about 60% iron and 1.5% copper. He will therefore discard no tailing, but make two valuable products, a magnetic that will carry 67% to 68% iron and a non-magnetic with 7% to 9% copper and generally high in iron.

Three years ago I shipped to Tacoma 1,100 tons from Tassoo Harbour, Queen Charlotte Islands, as average analysis of which was as follows:

Iron	62 %	Gold 0.02 oz. per ton.
Copper	1.8	Silver 0.4 oz. per ton.
Silica	3.6	
Lime	1.0	
Sulphur	2.8	

Based on an analysis, one ton of ore may yield 1,600 pounds of magnetite containing say 68% iron and probably less than 0.2% copper. The non-magnetic 400 lb. of copper ore analyzing about

Iron	38%	Gold 0.1 oz. per ton.
Copper	8	Silver 2.0 oz. per ton.
Silica	16	
Lime	5	
Sulphur	12	

[2]Trans. C. M. I. Vol. XVI, 1913.

This latter, sintered, will have a net value, for the 400 lb. of not less than $4.00, with copper at 15c per lb. and silver at 50c per oz.

The finely divided magnetite, sintered or briquetted, will be absolutely free from sulphur, will carry about 70% iron as sinter or 67% iron as briquette, and its physical character will command the very highest price in any iron ore market.

Such ore occurs at many points on the Coast, in apparently very large bodies, notably on Queen Charlotte Islands and Prince of Wales Island, as well as on Texada and Vancouver Islands. Generally its occurrence and location are so favourable that it may be mined, separated and sintered at a cost not to exceed $3.00 per ton—which means that 26 to 30 lb. copper per ton will pay all the expense, leaving our high grade artificial hematite 'on velvet.'

The conditions favouring the profitable establishment of an iron and steel industry on the Pacific Coast may be summarised, as follows: These are available:

Magnetic iron ore of superior quality at a cost f.o.b. barge, per ton	$0.75
Hematite second to none, f.o.b. barge, say	.15
Limestone of superior quality, f.o.b. barge	.50
Coke probably available at less than ex by-products.	5.00

As to labour, it can be trained; and as to a market it can be made and developed. The prospects in this respect are excellent.

As to electric smelting, without doubt any iron and steel enterprise should have an electric plant as an adjunct to produce high grade steels, but whether it will compete with the blast furnace in production of pig iron depends on the cost of power. The existing power companies if asked to quote a rate will doubtless give a tentative one of (say) one-half cent per kilowatt hour. On an exceptionally large contract, without long transmission, that figure might be cut in two, but probably a much lower figure would be requisite to ensure commercial success at normal prices. I offer for criticism an estimate on the cost of producing pig

iron at a site selected for its fuel supply. The latter is specified because of the fact that iron ore supply may come from various points, and it is desirable that coke be manufactured at the plant in order that the gases may be utilized. I assume that equal parts of magnetite and hematite will make a suitable mixture:

0.8 tons magnetite	$ 0.80
0.8 tons hematite	0.20
0.5 tons limestone	0.25
1.25 tons coke at $6.00	7.50
Freight on 2.1 tons	2.10
Labour	2.00
Interest and amortization	1.00
Power and sundry	1.00
	$14.85
Less by-products, including gas	2.00
	$12.85

This estimate is based on the assumption that iron ore can be supplied at the low figure of $2.60 per ton of pig iron produced. Of this $1.60 represents freight charges, which may be reduced considerably. Added to the estimate of the cost of coke of $7.50, which is probably high, the cost of these two items is $10.10, or lower than at Pittsburgh, where the iron ore required per ton of pig iron costs about $8.50 laid down, and where the cost of coke is not less than $3.00 or a total of $11.50. Limestone in sufficient quantity to flux the class of charge specified would not cost 40% of the figure given in the estimate.

Discussion.

Dr. J. G. Davidson: In regard to Mr. Hedley's reference to the by-product coking of coals, I may say that during the past two years we have been trying experiments on the recovery of marketable by-products, such as creosote and pitch, during the

coking of western coals; creosote being worth about fifteen cents a gallon. I am satisfied that the coke produced would be suitable for iron smelting, and consider that the Government should carry out experiments on the preparation of coke from 'Coast' coals in by-product ovens, and have the resulting coke tested in blast furnaces. We can also make charcoal of a fair quality from our western lumber, and at the same time secure valuable oils as a by-product.

Mr. E. A. Haggen: The problem to which Dr. Davidson referred, the production of suitable metallurgical coke from 'Coast' coals, has been solved by the manager of the Vancouver Gas Company, Mr. Keillor, who has been conducting experiments lately in the coking of Vancouver Island coal, with the result that, by the use of vertical retorts, he has obtained a coke of sufficient strength and freedom from deleterious substances, such as sulphur, to be adapted for metallurgical purposes such as iron smelting. The report of Mr. R. Lindeman, late of the Dominion Department of Mines, on the iron ores of the British Columbia coast was of a most satisfactory character so far as a supply of ores and cost of producing iron were concerned, but at the time the export was made there was not a sufficient market for the product. In 1909 and 1910, Eastern capitalists employed experts to investigate the problem of the manufacture of iron and steel in British Columbia, and their report was also satisfactory. The Vancouver Gas Company, however, did not believe it could supply metallurgical coke at less than $6 a ton. There is not at present any plant producing iron and steel on the entire Pacific coast of North and South America, with the exception of one small plant in California, which produces 12 to 15 tons of electric iron and steel a day. The Government should, therefore, be willing to assist in financing this industry, until it can be placed on a successful commercial basis, as a matter of national importance.

Mr. Nichol Thompson: Referring to the remarks of Dr. Davidson on the manufacture of coke and the recovery of by-products, I have sent the analyses of our Vancouver Island and Nicola Valley coals as well as samples of the Nicola Valley coal, to by-product experts in England, and their reports prove these

coals to be very high in by-products, which consist of light and heavy oils, creosote, tar and sulphate of ammonia. The value of the by-products was given as approximately $9 per ton, besides the coke, which would be valuable as a metallurgical coke, and would be exactly like what is costing $7 per ton in Sheffield, In fact the Nicola Valley coal is an ideal coal for distillation purposes and should yield a very big return. Concerning Mr. Hedley's remarks on the sintering of magnetite, and thereby converting it into hematite, I confess I cannot quite understand how this is accomplished or what advantage is to be gained by doing so, even if it can be done. I don't think it matters how the iron exists, as long as the ore is physically and mechanically suited for the blast-furnace, and I believe we have magnetite ores in British Columbia that are particularly well suited for use in blast-furnace without any considerable addition of softer ores. At one of the largest blast-furnace plants in England, they use principally the magnetite ores from Norway and Sweden with a mixture of spathic ores from Spain, and they manufacture all grades of pig iron from No. 1 to No. 4 or foundry iron. The following are typical analyses of these ores after drying:—

	Fe.	SiO_2	S.	P.	CaO	MgO	Mn
Highest Magnetite	69.3	2.42	.039	.014	.60	.52	.28
Lowest Magnetite	51.3	8.62	.277	.018	10.05	2.59	.29
Highest Spathic.	56.10	8.60	.30	.006	1.80	4.26	1.10
Lowest Spathic..	47.90	14.90	.286	.027	.80	.37	.76

Mr. Hedley's paper deals principally with magnetites containing copper, and of course those ores can only be made suitable for the blast furnace by magnetic concentration and briquetting. If the briquette can be made hard enough and porous enough so that it can stand the crushing strain and yet can be penetrated by the reducing gases in the blast furnace, that is all that is necessary, without any attempt being made to produce what Mr. Hedley calls artificial hematite. I have sent samples of our British Columbia magnetites to Sheffield and the report I got stated that high-grade tool steels could be made from these ores

at a cost of $2 a ton lower than from the best Norway ores on account of their high iron content, and lack of impurities; the following is a complete analysis, made in England, of some of our British Columbia magnetites:

Peroxide of iron	64.35	per cent.
Protoxide of iron	25.16	"
Protoxide of manganese	0.47	"
Alumina	0.96	"
Lime	1.00	"
Magnesia	3.89	"
Silica and insoluble	4.70	"
Combined water and moisture	Nil	
Arsenic and copper	Nil	
Sulphuric acid	Trace	
Phosphoric acid	Trace	
	100.53	"

Metallic iron	64.05 per cent.	This ore is of very good quality, indeed.
Sulphur	Slight trace	
Phosphorus	Slight trace	

I believe the above to be a fair average quality of our British Columbia magnetites.

There is a much better market for iron ore and products of iron and steel on the Pacific coast than is generally supposed. At the present time 82,000 tons of steel shipping is being constructed between San Francisco and Vancouver, and another 40,000 tons is under consideration. In a paper read before the American Institute of Mining Engineers in 1916, Mr. C. C. Jones[3] estimated the market for iron and steel products on the Pacific coast to be 923,000 tons per annum, adding that "probably the aggregate (consumption) is more nearly double this estimate." In 1912 British Columbia imported 120,000 long tons of iron and steel, including more than 7,000 tons of pig iron; and there is a much bigger market to-day for mine steel than there was in 1912.

[3]Trans. A. I. M. E. Vol. LIII, p. 308.

The production of pig iron in the United States has increased from 8,000,000 tons in 1896 to more than 37,000,000 tons in 1915. The total shipments of ore from the mines in 1915 exceeded 55,000,000 tons, and the increase in the production of pig iron amounted to 6,500,000 tons. In 1915, Great Britain imported from the United States 20,000 tons of steel billets valued at $11,000,000, and the returns for the month ending November 30, last, show that the unfilled orders of the United States Steel Corporation amounted to 11,058,541 tons, an increase of 1,043,282 tons over the preceding month. In 1912 Great Britain imported more than 6,000,000 tons of iron ore. The war will change the whole economic conditions of the Empire and if we are to maintain our position in the sun, and continue to be the foremost nation among the peoples of the earth, it can only be accomplished by conserving our natural resources and manufacturing them within the Empire for the use of the Empire first. Germany's strength in the present war lay in the provinces of Alsace and Lorraine, stolen from France in 1871. Out of 28,607,000 tons of iron ore extracted from German territory in 1913, 21,135,000 tons came from the Province of Lorraine alone. It behooves us, therefore, to see that our resources of coal and iron are not allowed to fall into the hands of foreigners who may again, as Germany is doing to-day, use them against us for our destruction.

Prof. John M. Turnbull: There seems to be a very general feeling that the time is rapidly approaching when an iron and steel industry can be established on the coast of British Columbia. There are many difficulties, but these can be summed up under three heads. The first is the difficulty in regard to markets for the products, which I believe can be overcome, because the markets will tend to develop as fast as the industry. The second point is the difficulty of smelting our ores, and this may also be overcome. The third point, the difficulty in regard to ores, is therefore the problem of first importance. I believe there are large quantities of magnetite ore on this coast, but we have no definite knowledge as to what quantity can be obtained and its probable average grade and character. The Provincial Government should make a geological investigation to determine clearly the character of the deposits and thus help in predicting their future possibilities.

Dr. Edwin T. Hodge: The greatest need of the Province of British Columbia at the present time is an inventory of its natural resources, and this can be accomplished by means of an intensive geological survey. The question of markets is of great importance. It has been stated that we can find a market in England for a considerable part of our iron ores and iron billets but there are many deposits of iron ore both larger and nearer to England than any we have here. The highly developed magnetite deposits of Gellivare and Kiruna, in northern Sweden, are fully as rich and very much nearer than ours. The Wabana deposits on the north shore of Bell Island, Conception Bay, Newfoundland, are very extensive and are highly developed. Other deposits available to the British Islands are those of Minas Geraes, Brazil, with reserves estimated at 2,000,000,000 tons; the Mayari and the Daiquiti, of Cuba, both with great reserves; and the Bilboa deposits of Spain. Another market which has been suggested is China. That country may be backward in some ways, but in the investigation of its natural resources it is far ahead of British Columbia. Its geological survey has shown that extensive deposits of iron ore exist. Since iron ore is usually carried as ballast, and since the rule is for the vessels to travel east in ballast across the Pacific, is it not possible that the near future will see China exporting iron ore? We hardly have a right to discuss questions of manufacture until we have settled the far more important matters of occurrence, quantity, availability, and smelting suitability of our iron ores. But in view of the great ease of transportation from Vancouver to many Asiatic cities, we ought with a proper supply of cheap iron ore to become a manufacturing and distributing centre for iron machinery or other high-grade products. It has been stated that several of our iron deposits are suitable for smelting, but as scarcely any of these have been opened we do not know to what geological type they belong and consequently cannot tell anything about their size or uniformity. We cannot, with advantage, build smelters until we have some definite idea as to tonnage nor can we do so until we know the type of ore we have to treat. I think you will all admit that with the limited geological examination which our iron ore deposits have received, except in a few instances, that we have only a faint idea as to the tonnage

and no conception as to the type of ores which a smelter would have to treat. Many of these facts, I am sure you will grant, must be known before we can plan definitely for a smelter and before we can interest capital in the manufacture of iron from our ores. Before we can interest capital in our Province we must make a geological investigation of our mineral resources. Our Province is so vast that the geological work which has been done means but little. The Geological Survey of Canada can only put three geologists into the field in this Province each year. If we are to know our resources and develop them rationally, we must also have our own geological survey. Our mining prosperity is dependent upon a careful and systematic investigation of our resources along geological lines.

METHODS OF MINING IN CROWSNEST PASS DISTRICT, ALBERTA.

By Raoul Green.

Rocky Mt. Branch, Nov. 1917.

The method of mining adopted in the Crowsnest district, situated along the Canadian Pacific Railway between Passburg and Crowsnest lake, depends to a great extent on the geological and physical features of the region.

Geological and Physical Features.—The main ranges of mountains are composed of Devonian limestone and have a general trend north-south. The overlying formation is Fernie shale which outcrops along the edge of the limestone; this in turn is overlain by Cretaceous rocks of which the most important member is the coal measures or Kootenay formation. These rocks, in general, form the foothills and are faulted parallel to the main ranges, forming a series of coal basins with the outcrops occurring along the crests of the hills. The Canadian Pacific railway follows a transverse valley cutting the coal measures at approximately right angles, thus affording facility for developing the coal seams at a number of different points. The mines are opened up by means of adits or tunnels driven from the level of the railway along the strike of the seam, and, as the tunnel advances, the distance to the outcrop increases so that it is possible to develop considerable coal without recourse to working 'to the dip.'

The following is an example of an actual developed coal basin:—

At A the east limb has a dip of 30 degrees to 45 degrees, but this gradually decreases till the lowest point of the basin C is reached; the western limb is sharply folded at B. This method of folding is repeated several times and most of the mines of this district are situated on the eastern limb. In a few instances slopes have been driven below the valley level towards the lowest portion of the basin, but although there is some decrease in the dip the workings have not yet extended to the western limb of the basin.

A variation of the above manner of folding often occurs when faulting takes place at C and the eastern limb is repeated as shown in the accompanying sketch. (Fig. 1)

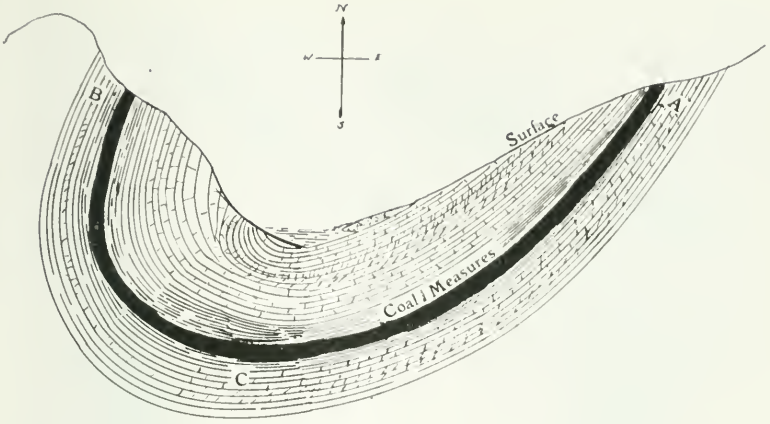


Fig. 1.

The coal measures average 400 feet in thickness and are overlain by a bed of hard conglomerate. Five seams have been found but with one or two exceptions one seam only has proven of present economic value. This seam is 12 feet in thickness and has a fair roof and floor so that conditions are suitable for mining. The seam produces some slack, but it is of good quality and makes a most suitable fuel for locomotive or stationary boiler use. Several of the transcontinental railways have already extended its use to British Columbia and the Pacific Provinces. The lump or screened coal is considered to be equal to any imported coal on the market.

System of Mining.—Two systems are used in the extraction of this coal, viz., up the pitch and across the pitch systems. In order to put either system into operation the method of development is the same namely:—main gangways, 500 to 600 feet apart are driven along the strike, airways are driven for ventilation at from 30 to 50 feet above these, and the rooms are opened off above the airways.

Up the Pitch System.—In this system rooms are driven from the airway, over the lower level or gangway, to the succeeding level above or to an airway beneath the level. These rooms are from 10 to 20 feet wide and the pillars left between rooms have a width of from 30 to 40 feet.

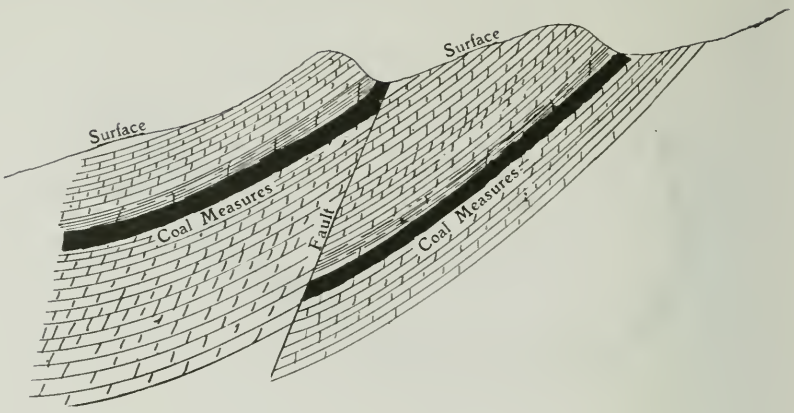


Fig. 2.

Roof conditions determine, to a great extent, both the width of rooms and pillars and the timbering method to be used. With a compact roof a single row of props is sufficient to maintain a room width of 20 feet but where the roof is soft it may be necessary to reduce the room width to 10 or 12 feet and substitute the square set method of timbering with lagging over the cap pieces and along the ribs. In this case the pillar width would be reduced also to about 20 or 30 feet.

In some cases where cap rock is encountered it may be mined separately and stored behind, but it is advantageous, wherever possible, to leave it in place, as well as any dirty coal adjoining the roof, and adopt a closer method of timbering.

In all cases where up the pitch system is adopted it is necessary to carry a chute up to the face of each workings for carrying the coal by gravity to the level below. The chute is constructed of boards one inch or two inches in thickness for the sides and two-inch plank for the bottom, lined with galvanized sheet iron of 22 to 28 gauge. One side of the chute is used for ventilation and as a travelling way. Figure 3 shows the method of timbering and chute construction.

Crosscuts, 6 feet by 6 feet, connecting the rooms, are driven every 50 or 60 feet to keep the ventilating current well up to the working face. After a certain number of rooms have been driven to the level or airway above, the extraction of pillars is begun. The pillars are drawn from the top downwards;

but, in a seam 12 feet to 14 feet thick, it is necessary to leave protection pillars in order to safeguard the miner against fall of debris from above. If the roof is soft, neither protection pillars nor any practical system of timbering is sufficient, so that this method has to be abandoned and the pillars split by driving a room up the centre of the pillar. By the latter method some 30% to 50% of the pillar coal is recovered. Where the pitch is over 35 degrees or the roof conditions unsuitable it is evident that this system is unsatisfactory as the recovery is too small.

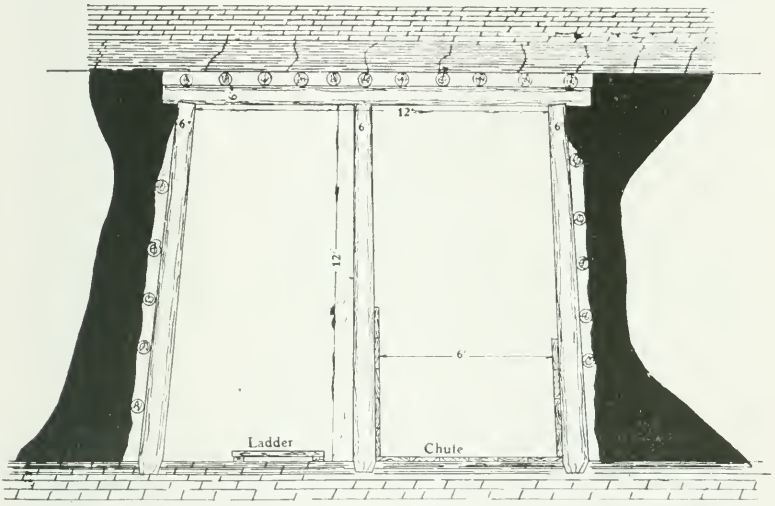


Fig. 3.

Figure 4 shows the 'up the pitch' system in use under good conditions; the pitch is 35 degrees and the roof is solid so that there is no difficulty in extracting most of the pillars. The shaded portions show the pillars extracted. The unshaded portion above No. 2 level has since been recovered and the coal dumped down No. 53 chute; the protection pillars were also recovered in a similar manner.

An adaptation of the 'up the pitch' system is often used where the dip is over 35 degrees This is called the angle system and differs from the first inasmuch as the rooms are driven at an angle to the pitch so that the rooms have a pitch of from 28 degrees to 30 degrees. The pillars are from 40 feet

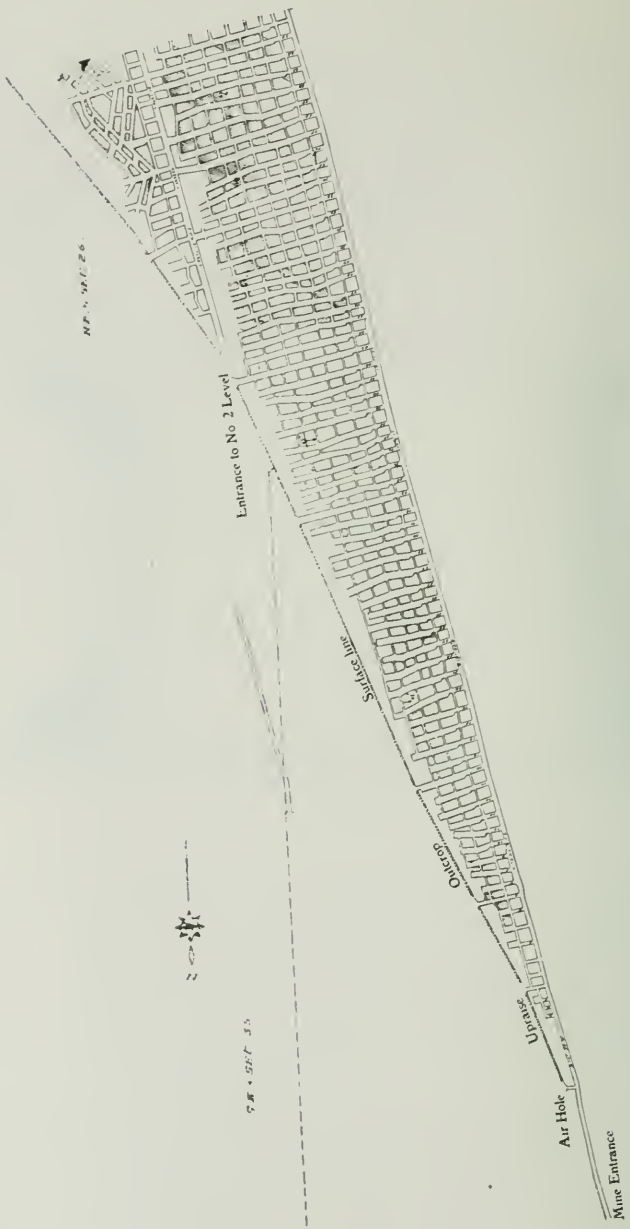


Fig. 4.—Horizontal Plane.

to 50 feet in width and from 60 feet to 70 feet in length. In the extraction of these pillars the workmen are well protected by leaving behind a small portion of a pillar to act as a guard against falls or caves from above. Under favourable roof conditions this barrier is blasted out and it is possible to recover over 80% of the pillar.

This system is also of special advantage in connection with the resumption of work after an extensive 'crush' as it is possible to close the rooms immediately below the cave, thus preventing the debris from following down and at the same time permitting of attack on the next pillar in the usual way. Where the roof is friable it may be necessary to split the pillars in the manner outlined above but by this method 60 per cent of the pillars may be recovered as against from 30 per cent to 50 per cent by the 'up the pitch' system.

Where the roof is hard and can be kept up by props, and the pitch from 30 degrees to 35 degrees, the up the pitch system is a cheap, safe and rapid method of mining coal; and the lower the pitch adopted in the rooms the greater the reduction in the breakage of coal.

Across the Pitch System.—In this system two parallel rooms are driven up the pitch to the level above. Rooms, 40 feet to 50 feet apart, are driven at right angles to them and continued to the next pair of parallel rooms, some 150 feet to 200 feet distant. The rooms are from 10 feet to 20 feet wide and the pillar width is from 20 feet to 30 feet; crosscuts for ventilation are driven every 50 feet to 80 feet apart. By this system the coal is loaded into small cars and trammed to the loading chute. For this reason it is not favoured by the miner so that its use is not general, which is regretted, as this system approaches the ideal one for working a pitching coal seam. Its advantages are as follows:—

1. In the case of a bad roof some method of timbering can be devised which will afford ample protection to the miner both in driving the room and extracting the pillars.

2. Where rock occurs in the seam it can readily be stowed behind.

3. When the room is finished the pillar can be drawn immediately. This is a great advantage as it causes a more uniform caving of the strata above, thus reducing the pressure which is a condition most desired in the working of a coal mine.

4. The ventilation problem is simpler as there is less difficulty in keeping the working faces well ventilated.

5. The worked out areas cave more easily; consequently there is less liability of the lodgment of gas.

Figure 5 shows the actual working of a coal mine by means of the 'angle system' and the 'across the pitch system.'

At A the angle system was used with good results; at B 'up the pitch' system was tried but as the roof was friable the pillars could not be extracted; at C the across the pitch system was tried with perfect results—practically all of the pillars being recovered right up to the surface. The same results were obtained at D. In this plan the shaded portions represent pillars that have been recovered.

Firing Shots and the Use of Explosives.—Where the coal is hard, blasting is resorted to. In some seams of the softer kinds the coal is mined by hand exclusively, no explosives being used. As a rule the face of the working places which are about 12 feet wide are undercut about 6 feet deep by 6 feet wide near the floor of the seam, and the holes are placed above this and the coal brought down; the rib coal afterwards is shot out toward the opening already made.

In some cases the undercutting may be done in the centre of the seam thus allowing the lower portion of the seam to be lifted up by shooting. These two methods are particularly applicable where there is a soft stratum of coal in the seam for 'mining.' Where the coal is hard and blocky it is often found necessary to shear the coal by making either a vertical cutting along the rib, or in the centre of the working place, the coal being shot to the shearing or cutting.

Permitted explosives alone are allowed, the one most commonly used consists of a mixture containing approximately 9% nitro-glycerine, 70% nitrate of ammonia, 15% potassium salts. All shots are fired by means of electric battery used by

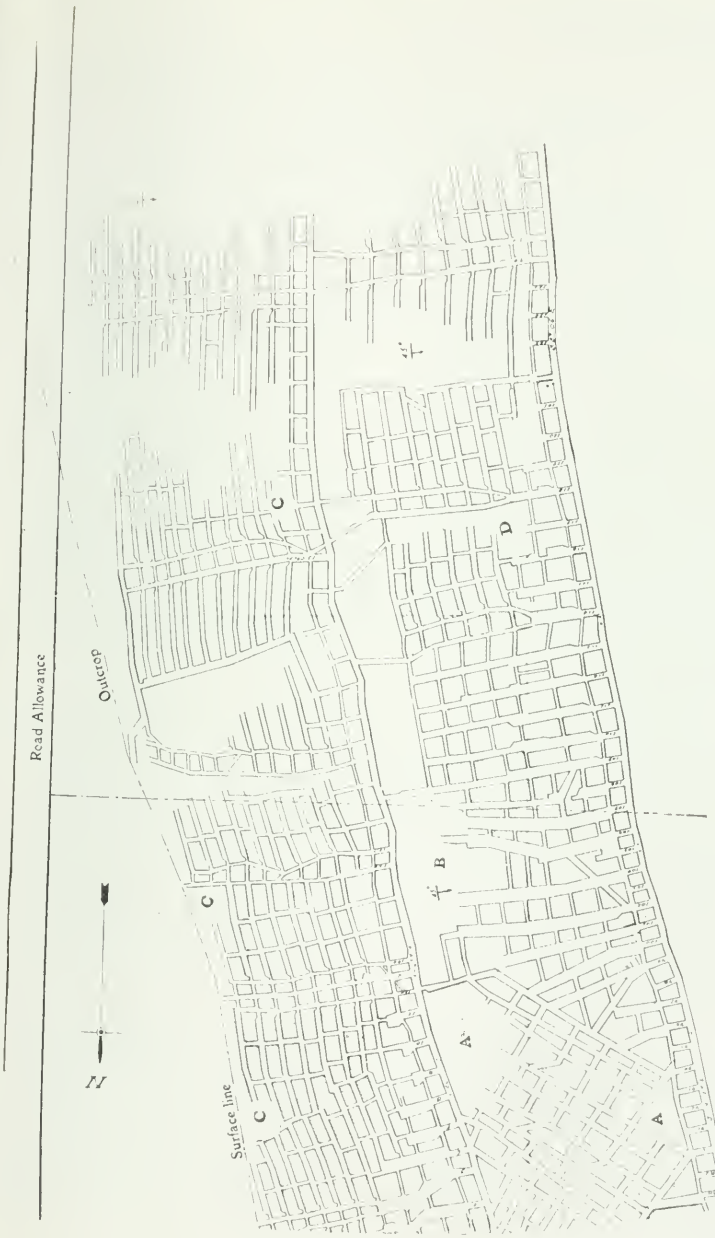


Fig. 5.—Horizontal Plane.

an official holding a certificate under the "Coal Mines Regulation Act."

The amount of coal extracted per pound of explosive varies from 4 tons in very tough and compact coals to 12 tons or more in seams where the mining is soft and the cleavage good. Owing to the heavy pitch of the seams in this district it has not been found advisable to use undercutting machines, the use of which could only be found of value where the coal is hard and compact; if some light machine, of the reciprocating type, could be devised, it would prove worth while investigating. In this paper no attempt is made to describe the haulage, ventilation or screening systems at the different mines; however, a few notes on ventilation may prove of interest. Where the coal to be extracted is above water level and where the outcrop can be followed for several miles along the hillside it has been found economical to move the fans so as to follow the advancing levels or gangways below. In this way the ventilating current is generated at a point as near as practicable to the working faces. By this method a smaller volume of air is required, less resistance has to be overcome and, therefore, less power is consumed. Further there is no necessity to maintain long airways with their heavy bills for repairs, etc.

The writer has in mind one instance of a coal mine having an output of 1,000 tons per day where the power consumption for ventilation is not over 25 H.P. and the water gauge $\frac{5}{10}$ of an inch. These results were obtained by the use of a fan of the multi-blade high-speed type, 4 feet in diameter and 2½ feet in width. This fan was easily set up or moved and was operated by an induction motor at 110 volts reduced from 2,300 volts—the voltage of transmission. The efficiency of the fan proper was 79%.

When workings to the dip become necessary the longer airways and larger volumes of air required will necessitate a different type of fan. This district has now come to the point where the upper workings are rapidly becoming smaller in extent and slopes are now being driven at several of the mines to develop the coal to the dip. This will involve the solution of new problems and will, unquestionably, increase the cost of production.

LUBRICATION OF MINE MACHINERY

By James A. Boyd, Edmonton, Alberta

Rocky Mountain Branch

The subject embraced by the title is a comprehensive one; I propose in this present paper to discuss certain phases of it only.

Steam Cylinders:—To lubricate a steam cylinder, the choice of an oil should be governed to a large extent by: (1) the temperature and pressure of steam, (2) physical condition of steam, (3) type of engine and valve and (4) location of lubricators. It is recognized of course that the oil should be atomized by the steam so that it goes to the cylinder as a spray or mist, and thus coats every part of the valves and cylinder. Hence good results cannot be obtained with high grade cylinder oil, when this is fed between the throttle and cylinder, or directly on the top of valves, even when operating under dry steam conditions, because the oil would thus be given very little chance to atomize, and if the flash point is too high, the tendency is for the oil to gum and to give uneven lubrication to the different parts. Such feeding also necessitates the use of more oil. Again too often the oil pipe is inserted into the steam pipe two feet or less from the cylinder and only protrudes into the steam pipe a short distance. As a result the oil drags along the side of the steam pipe and so does not atomize properly. Better results would be obtained if the oil pipe, when possible, were placed 6 to 8 feet from the cylinder. This would allow the steam more time to atomize the oil; while also less oil of a finer quality having a higher flash could be used. If a good cylinder oil adheres to the sides of the sight-feed glass of the lubricator, the difficulty can be remedied by lengthening the condensation connecting pipe. An atomizer can easily and quickly be made as follows: Insert the oil pipe into the steam pipe at least three-fourths of the way across. That is, if the steam pipe is 8 ins. in diameter, have the oil pipe project into it (say) 6 ins. Plug the end of the oil pipe and bore four to five holes in one side and place the pipe so that the small holes will face the same way as the steam is passing. This will tend to wipe off the small

drops of oil from the pipe, and also assist in drawing oil from the lubricator. The oil coming to the steam in small drops will atomize quickly and more thoroughly. The use of this atomizer, of course, is not always possible where steam is delivered vertically upward to the cylinders. Good judgement must be exercised, as it is sometimes impracticable to place this atomizer back from the throttle.

The physical conditions of steam referred to are moisture, and alkaline properties, often imparted to it by boiler compounds. It is known that mineral oil and water will not unite so as to form a mixture that will adhere to damp cylinder walls, and hence a small addition of fat is needed. Excessive moisture comes from over-taxed or priming boilers, or from dab piping. Sometimes this moisture contains foreign matter, other than alkali, which on entering the cylinder, adheres to the oil and becomes very hard under dry distillation. With over-loaded boilers, the moisture is apt to be delievered irregularly, especially if the load is fluctuating; or if piping is long, condensation is sure to be a steady factor to be reckoned with. Short piping may have bends and pockets in which water will collect and be delivered to the engine in sudden rushes. This action will explain why an engine will seem suddenly to lack lubrication at times, although most of time the lubrication may be excellent. A boiler compound used continuously in small quantities will cause little trouble, but whenever an extra amount is put in the boiler to remove scale, time is necessarily required to restore the boiler to a normal working condition. Soda ash, is a satisfactory boiler compound and is not as dangerous as some of the advertised boiler compounds, and is also much cheaper. The amount of cylinder oil to use depends largely, of course, on steam pressure, location of lubricator, length of stroke, and type of valve. Two drops should be sufficient ordinarily, and one drop in many cases is ample. The best results, in fact, are obtained by the use of the least amount of oil necessary for good lubrication. Putting a solution of salt or potash in the sight feed glass will increase the density of the water and cause the oil to flow in smaller drops; glycerine has the same effect. At coal mines are found many engines that carry a variable load; sometimes these are at less than 75% of their rated capacity, and with the pressure often falling below 85 lb. The

oil in these circumstances should be a compound filtered cylinder stock having a low flash point.

In the lubrication of external bearings, the method of supplying the oil is of greater importance. The old way was to oil by hand feeding. There are a number of self-acting oil feeding appliances which can be applied to the automatic oiling of old machinery originally designed for hand oiling. There is only one good way to oil any part of machine and that is by feeding the oil automatically. Hand oiling is in every way wasteful and unsatisfactory. The machine receives a feast or a famine of oil and most of the excess oil is wasted. Thurston says:" The oil-bath sometimes reduces friction to $\frac{1}{10}$ the amount observed with the hand oiler."

Not less important than the methods for delivering oil to a machine is the use of an oil having the right body. Fluid friction, that is oil friction, is very important in high speed machinery. A part of the oil between journal and bearing sticks to the journal and turns, and a part of it sticks to the bearing and does not turn. These two films of oil slip upon and along one another, and some friction is set up between them. The thicker the oil, the more and the thinner the oil, the less there is of this interior friction. Full lubrication is obtained by the use of the thinnest oil that keeps journal and bearing apart. The friction in the oil itself depends largely on speed. In slow speed machinery with high journal pressure, a heavier bodied oil should be used. The law of friction in the oil itself at high speed is: The friction increases as the square of the speed. Increase the speed by ten and you increase the friction by one hundred, increase the speed by a hundred, and you increase oil friction by ten thousands. At high speed, friction is almost incredibly great in amount. A large number of engine and machine oils sold by jobbers are doped to increase body and flash point. Emil F. Diedriche, a former large jobber and an authority on the compounding of oils and greases says: "To give oils a viscous consistency (i. e. to increase their viscosity) they are often compounded with proportions of oleate of lead (lead plaster) dissolved in the oils at a temperature of from 140° to 160°F. One to two per cent of unvulcanized rubber is also dissolved in the oil and stirred together under heat until

thoroughly diffused. Soap in various proportions and thoroughly dried is also compounded in petroleum oils by heating and stirring until freely dissolved. Lubricating oils have also been made by avaricous and ignorant compounders by mixing heavy resin oils with lighter petroleum oils, in the proportion of two to four lbs. to the gallon."

The viscosity or body of such oils, appears deceptively superior to some of the best lubricating oils, but as they form resinous deposits under the influence of frictional heat, they are entirely unfit for lubricating purposes, their effect being to gum up the bearings and friction is as a result greatly increased. This class of dope might be used for oiling slow speed machines without disastrous consequences, but one can readily see that with high speed or even medium speed machinery, an enormous increase in friction is inevitable. Therefore, in oiling shafting, engines, etc., use the self-oiling principle wherever possible, and an oil with the right body. An oil suitable for self-oiling bearings should be a straight mineral oil with a gravity of 24 to 29 and a flash test of 375 to 400 F.

Filtration of Oil.—A good engine oil can be filtered indefinitely, as many as 700 times. A very good filter can be made as follows: Lay a fold of cheese cloth over three inches of white cotton waste, well picked, this on fine felt about a half an inch thick, the whole on a perforated sheet or a fine wire cloth. The oil after passing through the filtering medium, drops into a funnel connected with a ½-inch pipe. At the lower end of the pipe there is a funnel-shaped flange of smaller diameter than the funnel at the upper end of the pipe. The small pipe is of such length that the lower end comes within one inch of the bottom of the tank or vessel having at least 14 inches of water. This kind of filter is easy to make and easy to clean. The cheese cloth gathers all the coarse dirt which can be shaken off, the cotton waste removes finer material, and the felt removes most of the remainder, and the final cleaning is effected by the water, yielding pratically pure oil. Heating the oil increases the filtering rate but does not yield a superior product. For this reason some of the oil filters on the market are equipped with a steam coil.

Lubrication of Compressors.—A pure mineral oil having a a high flash point is required.

Air-Driven Mining Tools.—A pure mineral oil having a low cold test should be used.

Cable Lubrication.—First treat with light bodied oil. This oil acts as a lubricant between the strands, preventing too great friction in the body of the cable. Next apply a coating of 'Anti-Corrosive Chain and Cable Coating.' This acts as a seal and prevents corrosion. After operating for some time, an application of black oil should be used. This will assist in reducing abrasion.

Pit and Mine Car Lubrication.—The proper lubrication of pit car journals is of more than ordinary importance and the writer has observed that it receives but scanty attention in some of the districts visited. The desirability of adopting a standard pit car journal is a matter worthy of attention. At present there is a wide variation in the types in use. There are thousands of pit cars in daily use in coal mining operations in Canada, and each car having four wheels, it will readily be seen that no inconsiderable part of the cost of maintenance is chargeable to pit car wheels. Nevertheless the lubrication of pit car journals is not done systematically and with the care and attention requisite to ensure the best results. There are a number of reasons for this; one of the most prominent is due to the present type of wheel and the poorly designed oiling device used to lubricate the journals. Much of the pit car oils used is wasted in the writers' opinion. This loss is fully 60% of the quantity used and this figure is probably often exceeded. The men, or frequently boys, employed at coal mines to look after the oiling of pit cars, seem to think that it is their duty to fill the cavity in each wheel with oil until it runs out in a small stream at one of the openings in the wheel cavity and then pass on to the next wheel with the oil can in an inverted position, leaving a trail of oil from one end to the other of the string of cars being oiled, the position of each wheel being marked by a pool of oil. Thus frequently, more oil is wasted than would oil every wheel and journal if applied in a proper manner on the bearing surfaces.

Briefly, mining cars may be divided into two classes: Those making use of the self-oiling principle, and those with bearings of the open type. For the former, winter and summer black

oils have been found satisfactory, or a cheap semi-fluid grease. In one case experiments with different kinds of oils and greases were made, and it was found that if the bearings were periodically cleaned thoroughly, the cheaper grades of oils and greases gave fully as efficient lubrication as the highest priced lubricants.

With reference to the latter type of cars, the use is recommended of an automatic greaser, consisting of a scalloped wheel or brush which revolves in a trough of grease and makes contact with the axle. With such a greaser, a special grease of a sticky nature, not costing more than three cents a pound, is advised.

Again, many mine superintendents could materially reduce the amount of car oil used, and at the same time secure equally as good if not better lubrication. This result could be achieved, by placing a competent man in charge of the oiling. Such a man who will more than earn his wages even if somewhat higher than the ordinary wages for his work. The monthly quantity of oil used should be recorded carefully and systematically and charged against the monthly production. The quantity of oil used per ton mined should be reduced until finally a safe working point has been established and therefore maintained under normal conditions. Of course, if the tonnage falls off to any considerable extent under established conditions, the car oil will not drop in the same ratio.

Storage of Lubricating Oils.—It is useless to maintain an oil-house, unless a record is kept of where the oils are being used, and in this connection it might be well to keep a record of the amount of oil required by the different oilers. Each man should receive a card upon which his receipts of oils and greases are recorded and a charge made against his particular station, that is, against machinery or equipment he oils. Each card serves for one month; at the end of the month all cards are turned in, and an estimate made of the quantity of oil requisitioned and on hand. Any wastage of oil by an individual oiler will soon become apparent, as will also any misuse or misappropriation. The moral effect on the oiler of this practice is also good, and results in making him less wasteful in his use of oil. For the recording of the daily oil consumption the appended form may be recommended.

DAILY OIL CONSUMPTION REPORT

Where used.....Month.....

Day	Cylr.	Eng.	Mach.	Dyn.	Blk.	Air	Ice	Sundry	Total	Cup Grs.	Gear Grs.	Total	Remarks
1													
2													
3													
4													
5													
6													
7													
8													
9													
10													
11													
12													
13													
14													
15													
16													
17													
18													
19													
20													
21													
22													
23													
24													
25													
26													
27													
28													
29													
30													
31													
Total..													

Signed..... Return to.....When complete.

PULVERIZED FUEL FOR LOCOMOTIVES

By J. S. Coffin, Jr.

Annual Meeting, Montreal, March, 1917.

The Canadian Department of Railways and Canals report that for the year ending June 30, 1915, the total mileage covered by a total number of 5,486 locomotives in use on Canadian railways was 116,354,044 miles. This represents an average mileage per locomotive per annum of 21,209 miles; or a rate of 58 miles a day, or only 2.4 miles an hour. It may be assumed, therefore, that at present over one-half of the time of locomotives is spent at terminals, attributable mainly to the necessity for cleaning fires, ashpans, flues and smokeboxes; to time occupied in inspecting and repairing draft, grate and ashpan appliances; and for firing up. Moreover, the delays to locomotives while waiting their turn to reach the ashpit track to receive attention as above mentioned, frequently exceeds the time that is occupied in the performance of the work itself; and during these delays much fuel is needlessly consumed, while, also, boilers are subjected to excessive contraction and expansion.

By the use of pulverized fuel for locomotives, ashpit delays are entirely avoided and this would result in considerably relieving the terminal congestion now existing. Hence the equipping of locomotives at present in use to burn pulverized fuel affords an easily available and rapid means for overcoming the existing serious freight congestion, and at the same time would enable Canadian railways to increase their earnings considerably.

A large percentage of Canadian coals are very friable. This precludes their successful and economical use when fired on grates in a locomotive firebox, but, as the results of the tests here presented demonstrate, these coals including lignite, when properly prepared in pulverized form, are capable of being utilized economically for the generation of steam as a substitute for the higher priced fuel oil and imported coals, and at the same time ensure smokeless, sparkless, cinderless and sootless operation, thus meeting public demand and legislative requirements with respect to smoke abatement and fire hazards.

History.—Although as early as 1818 experimentation with coal dust as a fuel was undertaken, no actual industrial application was made until 1895, when its use in cement works was introduced as a result of the high price of fuel oil.

During the past twenty years the Manhattan Elevated Railroad in New York City, the Swedish Government Railway, and some other steam roads have also experimented with the burning of peat and coal in pulverized form for locomotive purposes. So far as is known, however, the first locomotive using fuel in this form and successfully operated in regular train-service, was an engine of the 10-wheel type on the New York Central Railway. It was provided with a self-contained equipment for burning pulverized fuel in suspension. The engine was converted during the early part of 1914.

Since then locomotives burning pulverized fuel are in use on many of the large railways in the United States; and more recently they have been introduced in Brazil. Installations, in fact, are now being made as rapidly as the equipment can be produced.

Coal suitable for pulverized fuels.—From investigation of the various coals tested to date it may be affirmed that any solid fuel when in a dry pulverized form, having two-thirds of its content combustible, is suitable for steam generating purposes. Inferior grades of coal, anthracite culm, dust and slush, and bituminous and lignite slack, screenings and dust, as well as domestic and steam grades of anthracite, bituminous, semi-bituminous coals, and lignite and peat, are all suitable for burning in pulverized form.

Both time and money spent in briquetting small coal for use in locomotives is wasted for the most part, since the bonding material will always burn out first, and does not add appreciably to the fuel value of the coal. Moreover, the cost of briquetting at 75 cents to $1.00 per ton is prohibitive. The most efficient solution of the 'small coal' problem, therefore, will be obtained by the use of pulverized fuel.

Preparation.—In order to produce the best results the fuels must be dried mechanically until their moisture content does not exceed 1%. The required degree of pulverization will be

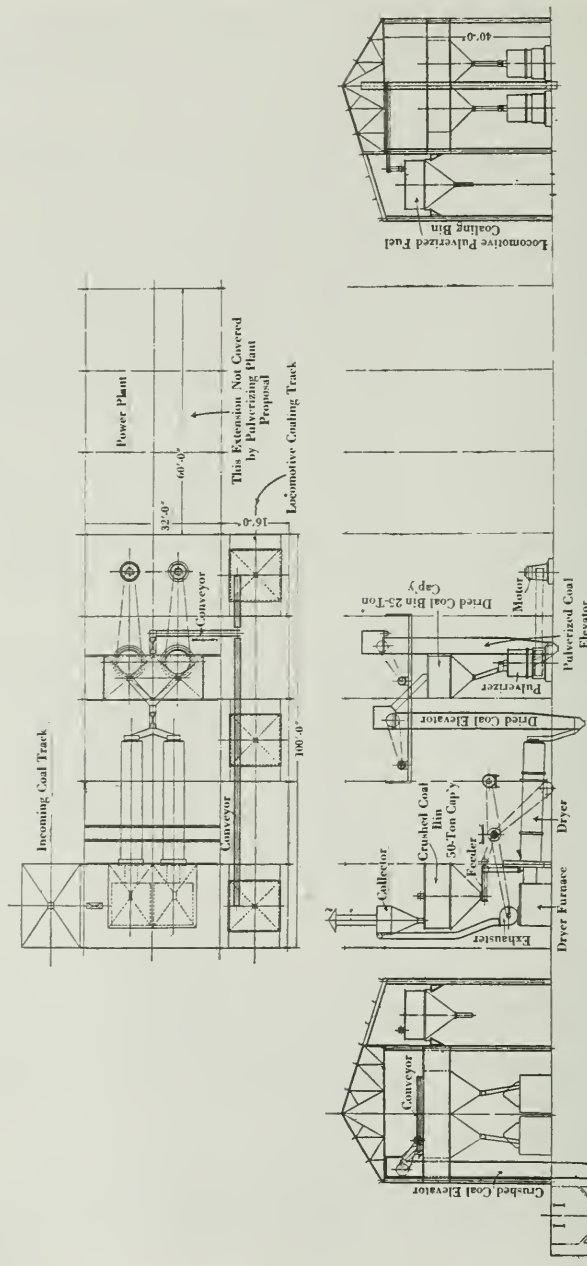


Fig. 1—Typical 15-ton per hour pulverizing plant and coaling station

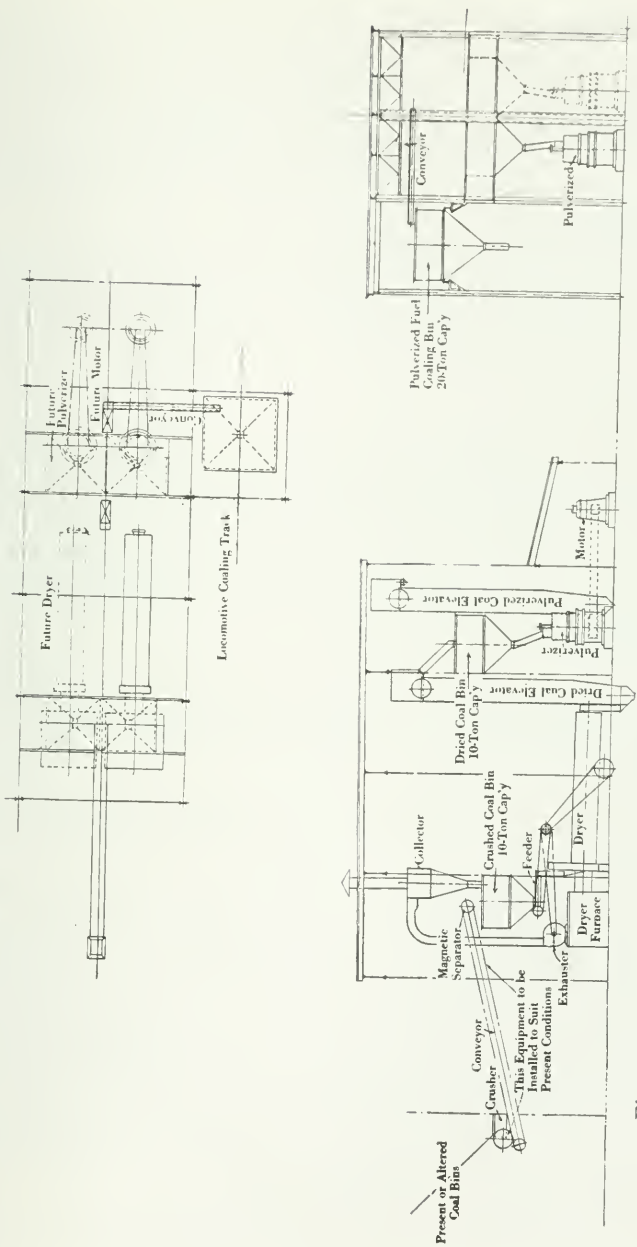


Fig. 2—Typical 4-ton per hour pulverizing plant arranged for eight tons per hour future capacity.

reached when 95% of the crushed coal will pass a 100-mesh screen, and at least 85% will pass a 200-mesh screen. This dryness and fineness is essential to insure every particle of the fuel burning instantaneously when in suspension. To accomplish this a suitable crushing, drying, grinding, conveying and storage plant is installed at the principal coal handling stations. (See Figs. 1 and 2).

Pulverized fuel should be stored in containers of metal or other fire proof material, and of the requisite capacity to supply the demand. Care must also be taken in the construction of these storage hoppers to prevent the prepared coal absorbing any moisture.

Coaling Stations.—The coaling station consists of one or more overhead storage bins capable of supplying 15 tons of pulverized fuel to a locomotive tender in three or four minutes without exposing it to the atmosphere, or producing dust clouds.

Positive control of the fuel flowing to and of the air thus displaced in the tender must be provided. Special equipment, designed to secure the foregoing requirements, must therefore be installed.

Equipment.—By referring to the accompan ying illustration in this paper (Fig. 3), a general idea will be obtained of the equipment that is used for burning pulverized fuel in a steam locomotive.

The following desiderata have been kept in mind in designing this equipment:—

(1) Readily applicable to either new or existing locomotives of standard design.

(2) Simplicity and the standardizing of various details, making them interchangeable for different types and sizes of locomotives.

(3) Applying all possible operating equipment in a self-contained manner to the tender fuel tank, leaving only the fuel and air supply control levers in the cab.

(4) The elimination of all complicated fuel conveying mechanism between engine and tender.

(5) Elimination of all manual handling of fuel, fire or ashes in the operation.

(6) Positive control over the fuel feed, to quickly meet all conditions of road or terminal operation.

(7) Provision for quick firing up, free steaming, perfect combustion, regularity of boiler pressure, uniform firebox temperature and maximum capacity of boiler with the minimum heat loss.

(8) The entire regulation of combustion is controlled by three levers in the cab, namely: fuel feed, air supply and induced draft (the latter is only used when locomotive is not using steam).

(9) The arrangement of the refractory material so as to provide ready accessibility to all parts of the firebox for inspection and maintenance.

(10) Positive assurance of a supply of dry fuel under all conditions of weather.

(11) Elimination of any need for firing tools, such as scoops, rakes, hoes, slash bars and grate shakers; as well as the elimination of cab glare, heat, and lowering of firebox temperature and draft resulting from furnace door openings.

(12) Reduction to minimum of noise and dust in the cab.

(13) The reduction of enginehouse facilities, delays, and expense incident to the building, preparing, cleaning and dumping of fires, and hostlering locomotives.

(14) Making the pulverized fuel burning and storage equipment on the engine and tender readily convertible for the use of fuel oil.

Operation.—In firing up a locomotive the draft is formed in the usual manner by turning on the stack blower. Then a piece of lighted waste is put through the firebox door opening and placed on the furnace floor directly ahead of the primary arch. The pressure fan and one of the fuel and air pressure feeders are then started. Full steam pressure can be obtained in from forty-five minutes to one hour from water at a temperature of 40° Fahr. The fire is then controlled by regulation of the fuel and air supply to meet the various conditions of standing, drifting or working. The stack blower is used only when the engine is not working steam.

The feeding and burning of pulverized fuel is briefly as follows:—

The pulverized fuel gravitates from the fuel tank on the

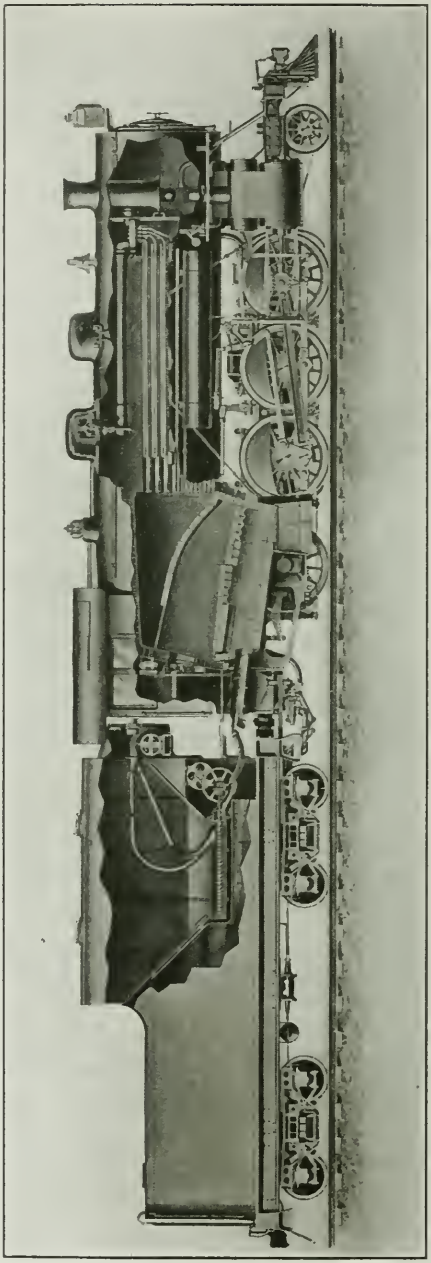


Fig. 3—Typical application of pulverized fuel burning equipment to consolidation type of steam locomotive.

tender to the conveyor screws and is carried to the fuel and pressure air commingler where it is thoroughly mixed with and carried by the 'pressure air' through the connecting hose to the fuel and pressure air nozzles on the engine. From these nozzles, it is blown into the fuel and air mixers where additional induced air is supplied. The combustible mixture, thus formed, is induced by smokebox draft into the furnace, where it ignites in nearly a gaseous form. The flame obtains its maximum average temperature, of from 2,500 to 2,900 degrees Fahrenheit, in the forward combustion zone under the main arch, where auxiliary heated air, drawn in by the smokebox draft, is added to finally complete combustion.

On account of the pulverized nature of the fuel, each particle is assured of being in direct and intimate contact with sufficient air to complete its combustion. The particles of white feathery ash left in suspension are either carried with the gases through the flues and out of the stack, or (when the temperature is higher than the fusing point of the ash) they are melted into slag which runs down the underside of the main arch and the front and sides of the furnace walls, into the self-clearing slagpan, where, by air cooling, it solidifies. The slag is readily dumped in the round house by means of drop bottom doors, thus eliminating ash-pits.

By referring to Fig. 3 it will be observed that the flame and products of combustion at all times flow by induction towards the stack, for as explained above, the fuel is blown, or forced, as it were, only as far as the mixing chamber, and induced from that point.

As a result, all destructive impinging action against the brick work and heating surfaces, is eliminated.

The fuel and pressure air feeder are so arranged that a certain number of revolutions represents a predetermined weight of coal; and as each feeder has a range in capacity of from 500 to 4,000 pounds of pulverized fuel per hour, and as it is possible to easily apply one to five of these to the ordinary locomotive tender, no difficulty is experienced in supplying the requirements of present day locomotives.

Danger.—Any combustible material or organic dust of a

given fineness, carried in suspension at a given temperature, is explosive.

Pulverized dry coal of the required degree of fineness is not dangerous unless in suspension, in nearly still air and mixed with the proper proportion of oxygen at the requisite temperature to produce primary ignition.

Where pulverized fuel has been stored and kept at a temperature below 150 degrees Fahrenheit in dry, air-tight and fire-proof bins, no cases of spontaneous combustion are known to have occurred.

Analysis of coal burned in pulverized form compared with analysis of Canadian coals in different provinces.—Tables 1 and 2 are of interest in comparing the analyses of various coals that have been successfully burned in pulverized form on locomotives in regular train-service with the analyses of representative coals mined in Canada.

Table No. 1

ANALYSIS OF SOME FUELS WHICH HAVE BEEN SUCCESSFULLY BURNED IN PULVERIZED FORM ON LOCOMOTIVES IN REGULAR TRAIN SERVICE

Name	Used By	On	Moisture	Volatile	Fixed Carbon	Ash	Sulphur	B.T.U
Atlas Portland Cement	N.Y.C.	Loco.	.44	26.27	62.00	11.43	1.35	
Illinois Unwashed Screenings	C. & N.W.	Loco.	3.70	35.25	48.74	10.61	1.70	12,400
American Loco. Co.	N.Y.C.	Loco.	.88	25.67	63.05	10.40	1.64	13,912
Central of Brazil	N.Y.C.	Loco.	1.73	14.02	60.04	25.94	8.25	10,904
Texas Lignite	D.&H.	Loco.	21.52	33.08	29.28	16.12	1.97	7,801
Eagle Pass	D.&H.	Loco.	.65	33.45	40.1	26.43	.84	8,845
Bankhead Coal	D.&H.	Loco.	.48	13.68	70.83	15.49	.94	
Canmore Coal	D.&H.	Loco.	.40	15.39	73.64	10.57	.90	13,950
Canadian Pacific Ry. Bienfait Lignite	D.&H.	Loco& Boiler	8.84	33.97	54.63	11.40		
(1) Run-of-Mine Bituminous	D.&H.	Loco.*	.50	29.50	60.00	10.00	1.50	13,750
(2) Hudson Coal Co., Anthracite Slush	Hudson CoalCo.	Boiler	1.00	6.00	71 00	22.00	2.50	11,250
Kentucky Unwashed Screenings	C.& N.W.	Loco.	2.46	36.00	54.00	7.94	.79	13,964

*A mixture of 40% of (1) and 60% of (2) were used.

Table No. 2

ANALYSIS OF SOME DIFFERENT COALS THROUGHOUT THE DOMINION OF CANADA

Mine	Moisture	Volatile	Fixed Carbon	Ash	Sulphur	B.T.U.	Equivalent Evaporation per pound	
							Unwashed	Washed
Nova Scotia								
Joggins	1.3	36.6	44.8	18.6	5.4	11,600	5.66	6.22
Springhill	2.2	33.3	63.3	3.4	1.0	14,190	6.63	7.36
Pictou	3.6	33.3	55.4	11.3	0.6	13,230		7.24
Inverness	9.3	40.0	49.6	10.4	6.0	12,150	5.46	5.67
CapeBreton or Sydney	3.5	37.4	55.4	7.2	2.9	13,770	6.74	7.01
New Brunswick								
Kings Mine (Queens County)	1.3	32.2	53.4	14.4	5.8	12,890	6.03	6.80
Saskatchewan								
Taylorton	28.6	42.9	49.0	8.1	0.6	10,690	3.91	
Estevan	30.9	40.0	43.2	16.8	0.5	9,650		
Alberta								
Strathcona Lignite	22.7	41.0	47.0	11.4	0.4	10,730		
Parkdale Lignite	22.5	37.8	51.3	10.9	0.4	10,910	4.98	
Standard Mine Lignite	23.5	42.0	49.9	8.1	0.4	11,360		
Taber Mine Lignite	13.0	36.0	49.9	14.1	1.4	11,040	3.91	
Lethbridge Lignite	8.4	37.5	51.5	11.0	0.8	11,710	5.92	
Coleman Seam 2	2.0	25.1	55.1	19.8	0.4	11,720	7.17	7.60
Coleman Seam 4	2.0	23.9	59.9	16.2	0.6	12,530		
Passburg, Leitch Col.	1.9	27.0	55.1	17.9	0.6	12,240	6.61	
Hillcrest, Hillcrest Col.	3.0	29.3	55.4	15.3	0.6	12,460	6.63	7.77
Hillcrest, Bellevue Col.	0.9	27.6	56.9	15.5	0.8	12,380	7.41	7.40
Canmore No.1	1.2	17.2	70.5	12.3	0.8	13,210	7.74	8.44
Bankhead	1.1	12.6	73.3	14.1	0.6	13,080	7.30	8.20
Bankhead Coal Dust	2.7	17.1	68.6	14.3	0.6	13,100	7.27	
British Columbia								
Michel No. 3 Mine	1.4	24.8	62.7	12.5	0.5	13,270	8.06	8.16
Michel No. 7 Mine	1.9	22.6	65.5	11.9	0.4	13,360	8.11	
Michel No. 8 Mine	3.0	24.1	65.7	10.2	0.6	13,480	8.24	
Hosmer No. 2 Seam	1.7	21.3	63.4	15.3	0.3	12,710	7.84	
Hosmer No. 6 Seam	2.6	25.6	62.0	12.4	0.6	13,090	7.57	
Hosmer No. 8 Seam	4.0	28.0	64.5	7.5	0.6	13,990		
Coal Creek No. 2 Mine	2.2	26.3	64.7	9.0	0.5	13,820	8.66	
Coal Creek No. 5 Mine	1.6	24.0	65.2	10.8	0.5	13,480	8.25	
Middlesboro Colliery No. 1 Mine	4.4	39.1	46.4	14.5	0.9	11,680	6.17	
Middlesboro Colliery No. 2 Mine	2.9	39.0	48.1	12.9	0.7	12,170	6.22	
Vancouver Island								
Wellington	1.8	40.1	49.8	10.1	0.4	13,160	6.76	
Douglas Seam	2.2	41.2	48.5	10.3	0.9	12,830	6.59	
Newcastle Seam	2.4	41.5	46.6	11.9	1.3	12,470	6.17	
Comox Seam	1.0	30.8	60.3	8.9	0.8	13,590	7.41	7.68
Suquash Seam	7.0	34.3	42.7	23.0	1.0	11,100	5.92	

These fuels differ more in physical characteristics than in chemical composition. The carbon and hydrogen content are usually taken into account for classification purposes, since the calorific value is mainly dependent on the percentages of these elements.

General Results.—When coal is fired in pulverized form at the rate of 3,000 pounds per hour, the smokebox gas analysis will average between 13% and 14% CO_2; when fired at the rate of 4,000 pounds per hour the CO_2 increases to between 15% and 16%, showing that there is no falling off in efficiency as the rate of combustion increases as is the case when coarse coal is fired in the usual way. Smokebox temperatures have varied from 430 to 482 degrees Fahrenheit, the temperature of the superheated steam varying from 580 degrees to 715 degrees Fahrenheit, respectively. The evaporation per pound of coal averages from 8.5 to 12 pounds of water, from and at 212 degrees Fahrenheit. The boiler efficiency, which on a superheater equipped locomotive boiler, working at from one-half to maximum boiler horsepower capacity, ranges from 65 to 55%, is increased from 10% to 20%. The sulphurous and other obnoxious gases resulting from combustion tend to combine with the liquid ash thus reducing materially the suffocating effect of the products discharged from the stack.

The following are average results obtained from some of the coals whose analyses are shown in Table 1.

TABLE 3.

LOCOMOTIVE PERFORMANCE—FUEL USED—KENTUCKY SCREENINGS.

Miles run	171
Running time, hours	3.87
Train, number of cars	5.8
Train, tonnage	291
Speed, miles per hour	44.2
Drawbar pull, pounds	2,711
Horsepower per hour	319.5
Fuel used, tons	3.82
Water used, gallons	8,381
Fuel per H.P. hour, pounds	6.17
Water per H.P. hour, pounds	56.48
Evaporation, water per lb. of coal, pounds	9.15
Evaporation from and at 212 degrees Fahrenheit, pounds	11.1
Boiler efficiency, percent	77

Table 3 gives average results obtained with an Atlantic type passenger locomotive burning pulverized fuel, rated at 21,850 pounds of cylinder tractive power, and having 81-inch driving wheels. It was used in high-speed passenger service, on round-trip runs of 171 miles in length.

The combustion results as indicated by the smokebox gas analysis were as follows:—

3,067 lb. of coal per hour $CO_2 = 14.5$; $CO = 0.0$; $O = 4.5$
3,498 lb. of coal per hour $CO_2 = 15.2$; $CO = 0.0$; $O = 2.8$
3,931 lb. of coal per hour $CO_2 = 15.2$; $CO = 0.0$; $O = 4.0$
4,000 lb. of coal per hour $CO_2 = 16.0$; $CO = 0.4$; $O = 2.6$

This locomotive performed the identical service of the regularly assigned Pacific type locomotives having 27,900 and 33,700 lb. tractive power and 69-inch diameter driver wheels, but burning coarse coal on the grates.

On D. & H. locomotive No. 1200 they are now burning dried pulverized anthracite sludge (*i.e.* the fine coal in suspension in the waters used for washing coal) mixed with 40% bituminous screenings. The wet sludge is diverted into bins where the liquid drains off leaving the fine coal. This is then dried and pulverized.

Exceptionally good results are being obtained with this locomotive, handling full tonnage over ruling grades; the evaporation being about 7 lb. of water from feed-water temperature per pound of coal.

Tests of pulverized Bankhead and Canmore coals, and Bienfait lignite, were conducted about a year ago on a consolidation type of locomotive, having 63,000 pounds tractive power, 63-inch diameter driver wheels, 27″ x 32″ cylinders, and carying a working pressure of 205 pounds. This locomotive is the largest of its type built up to that time. At the commencement of the tests the firebox brick work, air admission and draft adjustments were left as arranged for burning a mixture of 60% anthracite slush mixed with 40% bituminous screenings. With this adjustment Bankhead coal could not be burned satisfactorily on account of it not being possible to hold the fire when the locomotive was worked at full capacity. The heaviest pull made was two miles of 0.7% ruling grade with 1,577 adjusted tons.

As the limited amount of fuel for test would not permit making changes in the fuel burning and furnace adjustments, a test was made on October 9, 1916, with a mixture of 50% Bankhead and 50% Canmore coal. With this mixture the performance was much more satisfactory, although not what it would have been as regards free steaming or smoke elimination if the furnace had been specially adjusted for this fuel. The heaviest pull made was 34 miles with about $7\frac{1}{2}$ miles of 0.7% ruling grade and 2,302 adjusted tons.

During a test made on October 10, 1916, Canmore coal only was used and while the steaming was excellent the combustion was not smokeless. The heaviest pull made was 13 miles with about $7\frac{1}{2}$ miles of .7% ruling grade and 2,015 adjusted tons.

On October 11, 1916, the previous days' fuel and adjustment conditions were repeated with much improved combustion results. The heaviest pull made was 13 miles with about $7\frac{1}{2}$ miles of .7% ruling grade and 2,072 adjusted tons.

On October 12, 1916, a mixture of 90% Canmore coal and 10% Bienfait lignite was used with the same air admission and draft adjustments, but with a change in the furnace brick work to suit the much higher volatile fuel. The steaming was excellent and the combustion gave a clear and clean stack. The engine "could not be knocked out of steam" and after an $8\frac{1}{2}$-hour run about one peck of slag was found in the slagpan. The heaviest pull made was 11 miles with about $7\frac{1}{2}$ miles of .7% ruling grade and 2,189 adjusted tons.

On October 13, 1916, no run was made, on pending instructions regarding the use of the lignite which, when pulverized, contained about 10% of moisture. This adverse condition was due to the dryer in use not being adapted for this class of fuel. Although the moisture content was from 7% to 8% higher than that allowable for the best results with lignite, it was decided, on account of the previous days satisfactory results, to proceed with the tests.

On October 14, 1916, Bienfait lignite only, with 8% excess moisture was used, with the same adjustments as on October 12th. The steaming was not as good as desired, on account of improper furnace adjustments and excess moisture, but the

combustion was smokeless and the stack clean. The heaviest pull made was 15 miles with about 7½ miles of .7% ruling grade and 2,106 adjusted tons. After 13½ hours' run about 2 quarts of slag were found in the slagpan. No dust whatsoever could be felt as coming from the stack, which was absolutely clear of smoke, soot, cinders and sparks.

On October 15, 1916, a mixture of 50% Bankhead coal and 50% Bienfait lignite was used with the same adjustments as on October 12th and 14th. This mixture and adjustment gave the best results of any tried. The steam pressure ranged from a minimum of 195 pounds to a maximum of 205 pounds, with an average of 201 pounds. The locomotive was worked as hard as it ever has been over the heaviest pull and "could not be knocked out of steam" at a speed of 20 miles per hour. The smoke ranged from 0 to 1, as recorded by a Ringlemann chart, and at the end of a 6½-hour run about one quart of slag was found in the slagpan. The heaviest pull made was 11 miles with about 7½ miles of .7% ruling grade and 2,287 adjusted tons in thirty-nine loaded and five empty cars.

Conclusions from the Above Tests with Canadian Coals

Bankhead.—As this coal is extremely low in volatile matter, having only 10.44%, it is necessary to increase the volatile by mixing it with a higher volatile coal in order to get efficient and economical results.

Canmore.—With the proper adjustments of firebox brick work, air admission and draft, this coal, burned alone, will give very good results; but with a mixture of about 90% Canmore and 10% Bienfait lignite practically perfect results can be obtained. The addition of the lignite is desirable to increase the percentage of volatile matter above that in the Canmore coal alone.

Bienfait Lignite.—This lignite, properly dried and pulverized and burned in a firebox having correctly designed brick work, air admission and draft, will produce free steaming, and no smoke, soot, cinders or sparks in locomotive service.

General Conclusion.—The best results will no doubt be obtained from a mixture of the Bankhead and Canmore coals;

a mixture of either or both of the Bankhead and Canmore coals with the Bienfait lignite; or the use of straight Bienfait lignite for either locomotive or stationary boiler service.

STATIONARY BOILER RESULTS

By the use of pulverized fuel, the boiler pressure can more readily be maintained, increased, or reduced to meet the requirements, and when one or more of the boilers are not needed temporarily, the fuel can be stopped and started at will, thus eliminating the necessity of maintaining banked fires and burning fuel when not required, in order to have the boilers ready for instant use.

The following table indicates the results accomplished in the combustion of inferior grades of anthracite:

TABLE 4

POWDERED FUEL TESTS ON STIRLING TYPE BOILER

Test Number	1	2	3	4	5	6	7
Duration, hours	72	336	24	48	120	240	24
Horsepower rating	463	463	463	463	463	463	463
Horsepower developed, %	133	135	147	178	112	118	124
Fuel:—	Anth	Anth	Anth	Anth	Anth	Anth	Anth
Kind	B.E.	B.E.	B.E.	B.E.	Slush	Slush	Slush
Dryness, percent	0.65	0.65	0.65	0.65	0.8	0.8	0.8
Fineness, % through 200 mesh	86.0	86.0	85.0	86.0	88.0	86.0	88.0
Evaporation, from and at 212 degrees Fahr., lb.	8.7	8.9	9.6	9.8	7.8	8.1	8.5
CO_2, average %	16.6	16.3	15.9	16.6	16.2	16.5	16.7
Vacuum in breeching uptake inches of water	0.25	0.23	0.22	0.33	0.27	0.28	0.27
Vaccum in combustion chamber inches of water	0.16	0.14	0.13	0.16	0.17	0.19	0.15
Boiler pressure, aver. lb.	140	142	141	140	143	144	145
Flue-gas temperatures, deg., Fahr., average	518	525	496	603	475	580	576

The performance of anthracite birdseye hand-fired on grates, compared with the same fuel in pulverized form and burned in suspension, gives an average increase of over 40% in the evaporation in favour of pulverized coal.

In order to secure further data on the results to be obtained from the use of pulverized Bienfait lignite, a ten-hours' test was made on a 463 h.p. nominal rating Stirling type of boiler. This boiler, with cold water and brick setting, was fired up on October 10th, 1916, at 9 a.m., with pulverized bituminous coal. After 1½ hours 135 pounds steam pressure had been raised and the use of the lignite was then started and run for seven hours, after which anthracite slush was used for 1½ hours in order to note the comparative results. After 135 pounds pressure had been raised an average boiler pressure of 136 pounds was maintained, with an average boiler rating of 461 h.p. or 99.6%. The operation of the furnace was excellent, the stack clear and about one cubic foot of slag collected in the slagpan.

The test therefore shows that this lignite can successfully and economically be burned under stationary boilers.

ADVANTAGES

As already mentioned pulverized fuel provides smokeless, sootless, cinderless and sparkless operation of locomotives, thus meeting legislative requirements and public demand incident to fire and smoke.

The expenditure for locomotive fuel for steam railways in Canada now approximates nearly $30,000,000 per year, of which about one-third is expended to kindle, prepare, clean and maintain fires on grates when the locomotives are standing, drifting, or otherwise not actually using steam to move themselves either light or with trains.

By comparative tests made between similar locomotives in the same service, it was found that 2,775 pounds of lump coal, hand fired, was used to get up steam and for terminal handling and dead time, as against 1,569 pounds of pulverized screenings, or an increase of over 76%. In firing up alone the greatest saving results, as 1,700 pounds of lump coal were required as compared with 750 pounds of pulverized screenings, or an increase of over 125%.

The excessive rates of evaporation required to furnish the large quantity of steam demanded by the modern locomotive necessitates the utilization of heat units that now go out of the stack and into the ashpan when solid fuel is fired on the grates.

The proper burning of pulverized fuel in suspension represents the most advanced method for obtaining perfect combustion, with minimum heat losses.

A cubic inch of solid coal exposes only six square inches of surface, but when pulverized to the required fineness exposes about twenty-five square feet of surface. This enables faster and more uniform gas production from the volatile matter, and each particle of combustible matter is in direct and intimate contact with the required amount of air which results in complete combustion before reaching the flues.

There is no cooling of the fire by heavy intermittent charges of fresh coal and excess air entering the firebox through the fire door and grates, as is the case with hand or mechanical stoking on grates. The refractory material in the furnace retains the heat and prevents chilling of the firebox and flues when the fire is out. As practically dry fuel is fed into the firebox, there are no heat losses in vaporizing moisture. With powdered coal, there is no combustible matter left in the ash, and consequently there is a complete saving of the former ashpit loss—a loss which often runs from 2% to 7% of the total fuel fired either by hand or mechanica stokers.

RESOURCES, PRODUCTION AND CONSUMPTION

From the Government investigation of the coals of Canada, it is reported that the actual reserve for the different Provinces, is as follows:—

ACTUAL RESERVE

Province	Metric Tons	Class of Coal
Nova Scotia.....	2,137,736,000	Bituminous coal
Nova Scotia.....	50,000,000	Cannel coal
Saskatchewan.....	2,412,000,000	Lignite
Alberta.....	382,500,000,000	Lignitic or sub-bituminous
Alberta.....	1,197,000,000	Low carbon bituminous coal
Alberta.....	2,026,800,000	Bituminous and anthracitic or high carbon aituminous
Alberta.....	669,000,000	Semi-anthracite coal
British Columbia..	23,653,242,000	Semi-anthracite and bituminous
British Columbia..	118,000,000	Low carbon bituminous
British Columbia..	60,000,000	Lignite
Total.....	414,804,193,000	

Although Canada has large reserves of coal, the Department of Mines reports that during 1915 52% or 12,465,902 tons of the coal consumed in this country was imported, and of the 13,-209,371 tons produced, only 1,766,543 tons were exported.

The steam locomotives operating in Canada, consumed in 1915, 6,903,418 tons of coal, which is approximately 29% of the total consumption for the entire Dominion and about 53% of Canada's production.

By using Canadian fuels that cannot now successfully and economically be burned on grates, on account of their physical characteristics, but which can very efficiently be burned in pulverized form, not only will the natural resources of the country be developed extensively and increased tonnage produced, but the railways will materially benefit by the elimination of import duty and the reduction of high transportation charges.

Discussion

The President.—When we consider that of the total annua consumption of coal in Canada one-third, at present, is furnished to railway companies for locomotive fuel, the great possibilities of reducing the consumption of high-grade steam coal and of using instead, by the means Mr. Coffin has explained, products that heretofore have been regarded as valueless, for this purpose, will be realised. In this connection I should like to have Mr. Coffin's opinion concerning the practicability of utilizing peat in powdered form. I understand that on the Government railroads of Sweden pulverized peat is already being used successfully as a locomotive fuel.

Mr. Coffin.—Peat, I believe, is quite capable of being suc cessfully utilized as a locomotive fuel. It is very rich in volatile matter, as is well known. As yet, however, our experiments with this material are not complete.

THE FUTURE OF THE COAL TRADE OF THE EMPIRE.

By Allan Greenwell.

Annual Meeting, Montreal, March, 1917

One of the many interesting problems arising out of the War relates to the permanent or transitory character of industrial and commercial changes. Prominent among these changes is that which has taken place in international coal distribution.

During the period preceding the war, keen international competition was at work, made evident by tendencies rather than by marked changes in the coal export trade of the nations involved. Of the countries fortunate in possessing large coal resources, only a few exported coal to any considerable extent, but others were gradually coming into the market, while others again were dropping out. At the same time markets firmly held by one coal exporting country, were becoming disputed by others, as a result of general, industrial, political, and physical changes.

In a paper on" International Coal Competition," which I had the privilege of submitting to the Annual Meeting of this Institute in 1912, I showed that, in 1910, of the coal-producing countries of the world, only eight showed a balance of exports above imports.

Table A.

	Millions of Tons.
United Kingdom	$84\frac{1}{2}$
Germany	$20\frac{1}{4}$
United States	13
Japan	$4\frac{3}{4}$
Australia	$2\frac{1}{2}$
South Africa	$1\frac{1}{4}$
India	$\frac{1}{2}$
New Zealand (inconsiderable)	
	$126\frac{3}{4}$

Although Canada is not included in the above table, its omission is due to a peculiar circumstance, which was alluded

to recently in the Preliminary Report of a Special Committee of this Institute appointed to investigate conditions in respect of the Mineral and Metal Industries of the Dominion.

The Report states:—

"In 1913 the coal production of Canada was 15,012,178 tons, and the imports 18,201,953 tons; of the imports over 13½ million tons of bituminous coal was imported from the United States. In brief, although Canada possesses vast coal resources, the country now imports more coal than it produces. This is due in part to the fact that the coalfields of the Dominion are situated in the extreme eastern and western provinces. Conditions of transportation cost practically preclude the marketing of Nova Scotia coal at points farther west than (say) Brockville, Ontario, and similarly, coal mined in the Western Provinces is not marketable farther east than Winnipeg. The territory between these points, including the whole of Ontario and Manitoba is dependent therefore, on the United States for coal."

As a matter of fact, in 1913, Canada actually did export about 1½ million tons of coal, 80% of which went to the United States, and 14% to Newfoundland.

Since the war began, the coal production of Great Britain has been considerably reduced, chiefly owing to the large number of miners serving with the colours. In addition to this her increased requirements for naval purposes and the manufacture of munitions, and the fact that it has been necessary to supply large quantities of coal to her allies, has necessitated a strict limitation of exports, which have been reduced by about one-half. At the same time Germany's export of coal has been entirely limited to neutral countries in her immediate vicinity.

The partial withdrawal for a period of years from competition, especially in the more distant coal markets, of two countries which, in 1910, accounted for about five-sixths of the entire coal export trade of the world, has afforded an extraordinary opportunity for modification and re-arrangement of international relations in regard to the distribution of coal. For example, before the war, South America was supplied with coal chiefly by the United Kingdom and Australia. This was a growing market and was being assiduously cultivated. When

the export of British coal to that district was reduced nearly to a vanishing point, the carefully nursed market became the easy prey of the United States.

Opportunity of itself alone is not sufficient, however, to produce permanent changes, and it will be interesting to learn, as time goes on, which of the coal markets that have been won as a result of the exigencies of the war, will be retained when trade conditions become normal. Science has taught us that great changes are the result of natural forces acting gradually through long periods of time, rather than of cataclysmic action. The whole question of international coal competition is one of industrial and commercial equilibrium, and if any great national disturbance, such as the present war, produces a change of equilibrium which is not altogether stable, the irresistible tendency will be to revert to the *status quo ante*.

In the case of Canada, the total coal exports, which in 1913 amounted to 1,562,020 tons, fell in the following year to 1,423,126 tons, but the detailed changes are peculiarly interesting. In the advance chapter on "The Production of Coal and Coke in Canada during the Calendar Year 1914," which will form a part of the Annual Report on the Mineral Production of Canada, 1914, it is stated:—

"These figures show a decrease of 12.9 per cent in exports to the United States, which, however, with an importation from Canada of 1,088,983 tons, took 76.5 per cent of Canada's exports. Exports to Newfoundland showed a decrease of 20.58 per cent. Those to Great Britain showed an increase of 111.4 per cent, the total for the year reaching 25,576 tons. Under exports to other countries of 133,646 tons is included 40,978 tons to Australia, as compared with 13,889 tons in 1913." Unfortunately the figures for 1915 are not available, so that it is not yet possible to trace the progress of these tendencies under the present abnormal conditions.

With such coalfields as those of Nova Scotia and Alberta, situated respectively in contiguity with the coasts of the Atlantic and the Pacific Oceans, Canada would, at first sight, appear to be exceptionally well situated to carry on a considerable coal export trade. Whether this is true, when all the circumstances are taken into account, is a question for careful consideration.

It is significant that as recently as 1885 only five countries exported over 1 million tons of coal per annum, and that to the total the United Kingdom contributed about three-fifths, Germany one-fifth, Belgium one tenth, France one-thirtieth, and the United States only one-fortieth. Although by 1912 the coal production of the world had been trebled, and the total exports had increased at about the same rate, yet the United Kingdom and Germany still led as the greatest exporters, the former contributing about one-half, and the latter one-quarter, while the U.S.A. had advanced to one-eighth. At the same time Belgium and France had made little progress, but Japan and Australia had entered into the competition. These figures are shown in the following table.

TABLE B.

	1885		1912	
	(Millions of Tons)			
	Products	Exports	Products	Exports
Great Britain	159½	30¾	260½	86
U.S.A.	99	1¼	477	19½
Germany	58½	9¾	172	42½
France	19	1¾	39½	2½
Belgium	17½	5¼	22½	7
Austria-Hungary	8½	¾	15½	1½
Russia	4½	inconsiderable.	26 (1911)	inconsiderable.
Australia	3	inconsiderable.	11½	4
Canada	1¾	½	13	2
Japan	1½	½	17½	6
Total	372¾	50½	1055	171

Table C. gives a comparison of the Production, Import, Export, and Consumption of Coal per head of the population in different countries, in 1886-1890 (average of 5 years) and in 1912. From this table it will be seen that the great coal exporting countries are generally the great manufacturing countries. On this basis, however, it is difficult to understand why the per capita consumption of coal of Germany is not larger. The explanation may be found, perhaps, in the large consumption of home-produced lignite, amounting, in 1912, to nearly 80

TABLE C.

	Population		Production per head		Imports per head		Exports per head		Consumption per head	
	1886–90 000's omitted	1912 000's omitted	1886–90 Tons	1912 Tons	1886–90 Tons	1912 Tons	1886–90 Tons	1912 Tons	1886–90 Tons	1912 Tons
United Kingdom	36,870	45,315[1]	4.60	6.00[1]	0.00	0.00[1]	0.94	1.92[1]	3.66	4.08[1]
India	.....	291,180	.....	0.05	.....	0.00	.....	0.00	.....	0.05
Canada	4,698	7,490	0.49	1.73	0.45	3.32	0.11	0.24	0.83	3.32
Australia	3,003	4,636	1.15	2.53	0.00	0.00	0.37	0.82	0.78	1.71
New Zealand	610	1,037	0.96	2.10	0.18	0.35	0.10	0.22	1.04	2.23
South Africa	.....	6,202	.....	1.17	.....	0.01	.....	0.23	.....	0.95
Russia	104,100	173,320[1]	0.05	0.15[1]	0.01	0.04[1]	0.00	0.00[1]	0.06	0.19[1]
Sweden	4,325	5,917	0.04	0.06	0.28	0.91	0.00	0.00	0.32	0.96
Germany	47,900	64,434	1.32	2.59	0.08	0.17	0.22	0.64	1.18	2.12
Belgium	6,019	7,560	3.11	2.99	0.19	1.31	1.02	0.95	2.28	3.35
France	38,620	39,745	0.57	1.00	0.27	0.54	0.02	0.06	0.82	1.48
Spain	17,280	20,028[1]	0.06	0.18[1]	0.09	0.13[1]	0.00	0.00[1]	0.15	0.31[1]
Austria-Hungary	40,650	49,734[1]	0.22	0.31[1]	0.08	0.23[1]	0.01	0.52[1]	0.29	0.52[1]
Japan	40,580	51,025[1]	0.05	0.34[1]	0.00	0.00[1]	0.02	0.10[1]	0.03	0.24[1]
United States	59,980	95,444	2.06	5.00	0.01	0.02	0.03	0.20	2.04	4.82

[1]These figures refer to 1911. In the case of the United Kingdom the figures were abnormal owing to the national coal strike.

In the other cases, the figures for 1912 were not available when this paper was written.

Quantities less than 0.01 are shown as 0.00.

million tons, or about $1\frac{1}{4}$ tons per head of the population. The production and consumption of lignite in Austria, in 1912, was nearly 26 million tons; in Hungary, in 1911, about 8 million tons; and in Spain, in 1911, about $\frac{1}{4}$ million tons. In the cases of the United States and Russia, the returns include lignite.

The periods 1886-1890 (average of 5 years) and 1912 were selected as being the earliest and latest for which the records are comparatively complete.

Great Britain is, and always has been, by far the largest coal exporter. She produces about one ton per head more, and consumes about $\frac{3}{4}$ ton per head less than the United States. The imports in both cases are relatively negligible. During the period from 1886-90 to 1912, the production of the United States per head increased by 150 per cent, while the consumption increased nearly at the same rate, the difference being accounted for by an increase of the rate of export by nearly 600 per cent. It must be noted, however, that the export at the latter date amounted to only one-fifth of a ton per head, so that the increased production was practically absorbed by increased consumption. On the other hand, during the same period, the rate per head of production of the United Kingdom increased by 130 per cent, while the consumption increased by only $11\frac{1}{2}$ per cent. At the same time the rate per head of export increased by 104 per cent, so that the increase has been absorbed chiefly by increased export.

As the history of the British coal export trade, so vividly outlined by Stanley Jevons in "The Coal Question," throws a powerful light upon the subject of coal export generally, a short summary of its salient points will be useful. It is usually admitted that living beings of the same nature and in the same circumstances tend to multiply in the same geometrical ratio, so that if the population of a country increases at a diminishing rate, does not increase, or actually decreases, it follows that conditions adverse to its natural progress have set in. A necessary condition of the continuance of a normal rate of increase in the population of a country is a corresponding increase in the supply of food, and if the country is dependent upon itself for that supply, a limit must at last be reached dependent upon the area of the country suitable for cultivation.

According to the best statistics available, the population of England and Wales in 1570 was a little over 4 millions. During the next 60 years the population increased at the rate of 5 per cent for each period of 10 years. During the next 40 years the rate of increase fell to 1 per cent. During the next 41 years, the rate was .2 per cent. During the next 10 years there was no increase; and during the next 10 years, there was actually a rate of decrease of 1 per cent. After that time the population steadily increased at an increasing rate of increase, which reached a maximum figure of 18 per cent, for the 10 years 1812-1821, after which it has steadily decreased to 11 per cent, for the 10 years, 1902-1911. The middle of the eighteenth century constitutes a period of radical change in the history of British industry. Before that period the country abounded in corn, wool, meat, timber, and raw materials, which it exported. Since that period Britain has imported the very merchandise which she formerly exported, and has exported in their place articles manufactured from the raw material imported, and coal. In this connection, manufactured articles must be considered as containing, potentially, the quantity of coal directly and indirectly used in the numerous processes of their manufacture.

About the middle of the eighteenth century, under the conditions then prevailing, Britain had arrived at the maximum limit of population that could be supported by agriculture. From that epoch onwards, can be traced the gradual recognition that coal and not the products of agriculture was the staple industry of the Kingdom. In Jevon's words:—

"For the present (1865) our cheap supply of coal, and our skill in its employment, and the freedom of our commerce with other wider lands, renders us independent of the limited agricultural area of these islands, and apparently takes us out of the scope of Malthus' doctrine. We are growing rich and numerous upon a source of wealth of which the fertility does not yet apparently decrease with our demands upon it. Hence the uniform and extraordinary rate of growth which this country presents. We are like settlers spreading in a rich new country of which the boundaries are as yet unknown and unfelt."

In 1913 the aggregate value of the merchandise imported by the United Kingdom was nearly 769 million pounds sterling,

of which food and raw materials accounted for about 600 million pounds. On the other hand, the exports amounted to about 635 million pounds sterling, of which coal alone was valued at about 146 million pounds, exclusive of freight.

It is easy to see why Britain's rate of export and consumption of coal per head is so high. The exports of coal are employed in the purchase of food and raw material for manufacture; while the consumption of coal includes the equivalent of coal contained in the manufactured raw material which is exported in exchange for food and more raw material.

Turning to the United States, it is found that in 1913 her exports of merchandise exceeded her imports by 653 million dollars; and that the export of coal was comparatively inconsiderable. In this case the high rate of consumption of coal per head of the population was principally accounted for by coal used in manufacture employed directly or indirectly in the internal development of the country.

In the case of Canada, although possessing enormous coal reserves of her own, she imported from the United States, in 1914, 54% of her total consumption. The reason for this has already been given. The consumption of coal per head was 3.325 tons, a rate of consumption exceeded only by the United States, the United Kingdom, and Belgium. The total annual value of Canada's production, derived from Agriculture, Live Stock, Fisheries and Forestry, so greatly exceeds her own requirements that in 1914-15 she was able to export the following values of merchandise:

Table D.

Agricultural produce.....	$134,746,050
Animals and produce.....	74,390,743
Fisheries and produce.....	19,687,068
Forest produce.....	42,650,683

Canada is therefore not only able to supply a rapidly increasing population with home produced food, but has at the same time an enormous surplus for trading purposes. This surplus is readily taken by countries, less happily situated in

regard to food stuffs, and manufactures are returned in exchange. Thus Canada can trade with material which is annually being renewed, while Britain is compelled to export a national asset which cannot be replaced.

It would therefore seem that a country situated like Canada, should not dream of encroaching upon her valuable coal reserves for the purpose of developing an export trade in that substance, so long as there is an eager market for her reproductive produce, and so long as she is able to contribute to that market out of her surplus production. The time will no doubt arrive when Canada will, like Britain, be only too thankful to be able to fall back upon her coal resources for the purposes of international exchange, and will then grudge every ton of coal which has been disposed of prematurely. The question of coal production for home consumption, especially in connection with internal development, is another matter altogether.

The position of Canada in the following table, giving the world's production of wheat (in millions of quarters of 480 lb.) for the Season, 1915, will add emphasis to the foregoing remarks.

TABLE E.

United States		126
Russia		113
India		48
Canada		48
France		32
Italy		21
Hungary		20
Argentine		24
Germany	(say)	17
Spain		17
Rumania		11
United Kingdom		9
Austria	(say)	8
Australasia		4
Other small producers		50
		548

It is submitted that the same course of reasoning is applicable, *mutatis mutandis*, to the rest of the British Dominions, as well as to all other countries. It is a sound economic law that a country should export that which it can most easily produce and replace, and that coal, which is accumulated energy, and which cannot be replaced, should only be exported as a make-weight, and as a last necessity to make up the balance of trade.

The abnormal conditions produced by the present war have tended to disturb the tenour of trade exchange, but it is believed that the effect of these tendencies will be only temporary, and that a reversion will ultimately be made to the principles upon which international trade relationship has steadily developed.

Discussion

Mr. W. J. Dick—At the annual meeting of the Canadian Mining Institute in 1916 the writer presented a brief paper on the "Coal Situation in Canada" which, in part, may be summarized and supplemented as follows:—

The coal deposits of Canada compare favourably with those of the greatest coal mining countries of the world in respect of quality, quantity and accessibility for mining purposes, but, owing to their location in the eastern and western portions of the Dominion the large central market is supplied by imported coal.

United States bituminous coal is used in the area between a north and south line through Farnham, Que., and a line drawn from Battleford to Moose Jaw and thence to Estevan, Sask. Although a considerable quantity is used in Manitoba and Saskatchewan, these provinces are also supplied by coal from the Crowsnest, Canmore, Edmonton, Lethbridge and Souris districts.

Eastern Canada possesses no deposits of anthracite coal, and as this class of coal is admirably suitable for domestic heating and cooking purposes, it is imported in considerable quantity from the United States and is sold over an area extending from Nova Scotia in the east, to Battleford, Sask., in the west. The

imports in 1913 amounted to over 4,640,000 tons, being more than double the imports of 1906. From this it can be seen that the demand for this class of coal is increasing, notwithstanding the increasing prices; again, the supply of anthracite coal in the United States is limited and there is no assurance that its export to Canada will be long continued. In 1913, it was estimated that there were 16,153,000,000 tons of anthracite coal in the United States. In 1913, 91,524,922 tons were mined and, as it has been estimated that for every ton of coal lost a ton and a half is sold or used[1], the exhaustion is proceeding at the rate of 152,541,536 tons per annum. If the production continued at the same rate it would exhaust the anthracite of the United States in little over 100 years. We must, therefore, expect that the price will gradually increase until only the wealthy few can afford it. Coincidently with the rising price, production will decrease, thus prolonging the life of the mines.

Of the total consumption during 1913, 42.6 per cent was domestic coal and 57.4 per cent imported coal, or in other words we imported more coal than we produced. The importance of this fact may be recognized more fully when it is realized that the value of the coal production in 1913, greatly exceeded that of any other mineral and amounted to over 25 per cent of the total mineral production of Canada for that year, being valued at about $37,335,000. The situation then is this: although Canada has over 17 per cent of the world's reserve of coal, our production is small and we import more than we produce.

It is desirable both from the mining and national standpoint that these conditions be changed. Has anyone stopped to think what would happen if United States for one reason or another was unable or should prevent the exportation of hard and soft coal to Canada? We had, last winter, a very slight indication what it would mean and some of our educational institutions were forced to close their doors because of the shortage of coal due to a temporary freight blockade. This question may be resolved into these problems:

(1) Domestic fuel problem in central Canada.

[1]Mineral Resources of the United States—Part II, 1913, p. 728.

(2) Imported bituminous coal fuel on railways in Central and portions of western Canada.

(3) Domestic fuel problem in Prairie Provinces.

(4) Cheap power problem in Prairie Provinces.

(1) The following solutions of the domestic fuel problem in central Canada are suggested: (a) By the installation of by-product coke ovens at certain points on the St. Lawrence and Great Lakes system, the coke being used for domestic purposes in place of anthracite coal. (b) By the development of a peat industry in certain well chosen localities. (c) Eventually, no doubt, electric energy will, to a certain extent, replace coal for heating purposes in this area.

(2) A suggested economic solution of the railway fuel problem in central Canada is the electrification of our railways which, for economic reasons, could be undertaken step by step. In so far as the railway fuel problem in certain portions of the Prairie Provinces and Western Canada are concered, this may be solved by coal dust firing; (using western lignites or sub-bituminous coal) use of a suitable briquetted fuel made from lignites or bituminous coal, and the further use of our own bituminous coal.

(3) and (4) Two problems of great importance exist in the Prairie Provinces today, and their solution will become a matter of even greater moment in the future. These are: the problem of cheap power and the problem of a domestic fuel supply. In western Manitoba, in Saskatchewan, and in eastern Alberta water-power development costs, are, in most instances, high, but these districts are within reach of great deposits of lignite. It is, therefore, of great importance, that something be done with a view to utilizing the low-grade fuels which underlie the greater portion of Alberta and part of Saskatchewan and Manitoba. In order to make the coal transportable and suitable for domestic and power purposes it would be necessary:

1. That it be of sufficient value to be able to bear the cost of transportation.

2. That it withstand handling and a certain amount of weathering.

3. That it be a suitable fuel for domestic and power purposes.

Coal briquettes and carbonized lignite briquettes fulfill these conditions.

The Mines Branch, in co-operation with the Advisory Council for Industrial and Scientific Research, is working earnestly on the first of these problems and indications point towards an economic solution.

BITUMINOUS SANDS OF NORTHERN ALBERTA.

By S. C. Ells.

Annual Meeting, Montreal, 1917

The three outstanding features presented by a consideration of the deposits of bituminous sands of northern Alberta are:

1. That the deposits represent the largest known body of solid asphaltic material;

2. That these deposits are, as yet, totally undeveloped;

3. That at the present time practically all asphaltic materials used in Canada are imported from foreign countries.

There is a reasonable probability that, sooner or later, conditions will permit of a considerable commercial development of our Albertan deposits. Assuming this to be so, it is of importance to determine how best this development can be undertaken and an industry established on a sound economic basis.

Up to the present time, practically all information regarding the Albertan deposits, as well as the work that has been done in investigating the economic value of the bituminous sand itself, may be attributed to the Canadian Department of Mines, and, under existing conditions, it was but reasonable that a department of the Government should undertake this initial

---

Editor's Note.—Geographically, the Alberta deposits of bituminous sand lie approximately 300 miles to the north of the city of Edmonton which is, at present, the nearest centre of population. The transport of the material by water is impracticable, but rail facilities will presently be available, upon the completion in the early future of a line known as the Alberta and Great Waterways Railway. Topographical maps of the area, prepared by Mr. Ells indicate that the northern terminus of this road will be within reasonable distance of certain of the more important outcrops. The actual extent of the deposits is very large, and outcrops of the main body of bituminous sand extend along the various waterways throughout an aggregate distance of upwards of 180 miles. The preliminary report on the Bituminous Sands of Northern Alberta by Mr. Ells and also his various contributions to the Summary Reports of the Mines Branch for 1914-1915 and 1916 contain a great deal of detailed information regarding the various factors which will govern commercial development of the deposits. During the past year at the Mellon Institute of Industrial Research at Pittsburg, Pa., Mr. Ells devoted much time to a study of methods applicable to the commercial separation of refined bitumen from the crude bituminous sand. He has also conducted experimental research looking to the commercial distillation of petroleum from the crude bituminous sand. The results of this important investigation have not yet been published.

work. Practically the entire area is held under government reserve and consequently there has been little inducement for private individuals or companies to interest themselves in what has proved to be a somewhat extensive investigation.

Apart from the Canadian occurrence, bituminous sands have also a fairly wide distribution throughout the United States. A very brief reference to these latter occurrences, and to the history of their attempted development, will obviously be of value in considering the possible future development of the Albertan deposits.

The principal occurrences of bituminous sand in the United States are found in Kentucky, Oklahoma, California, and Utah. Extensive deposits of bituminous limestone occur in Texas and in Oklahoma. Most of these deposits are, however, relatively small compared with those of the Province of Alberta. Although the material from the various deposits in Canada and the United States is of the same general character, yet there are many varieties, each of which requires different manipulation.

In the United States, development of deposits of bituminous sand may be considered under two heads:

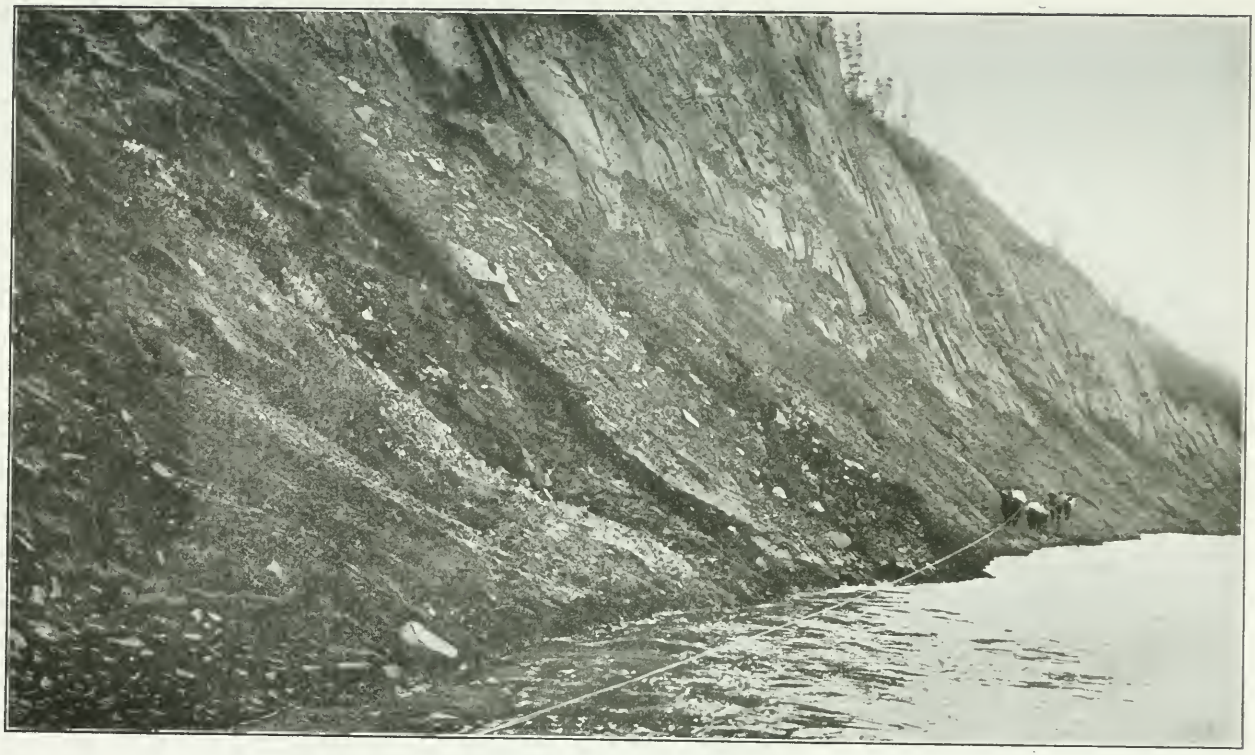
1. Separation or extraction of the bitumen from the crude material in order to secure a more or less pure product; and

2. Use of the bituminous sand as a paving material.

An indication of the success that has marked this attempted development in the United States is seen in the production that has been attained at the end of twenty-five years. In spite of the admittedly very large tonnage of bituminous sand available, the quantity[1] mined during 1913 was but 57,549 tons, valued at $173,764. During the same period, the production and importation of other asphalts into the United States, amounted to 648,459 tons, valued at $5,483,737. The value of the bituminous rock was thus slightly over 3% of all asphalts produced in or imported into the United States during that year.

Regarding the history of separation or extraction, practically no written records are available. Whenever separation has been attempted, either hot water at 200° F. to 212° F., or some solvent, usually a petroleum distillate, has been adopted as the active

[1]Mineral Industry, 1914.



Exposure on east side of Athabaska river, immediately below Mountain rapids. The bituminous sand here attains a thickness of over 150 feet, but the overburden is very heavy.

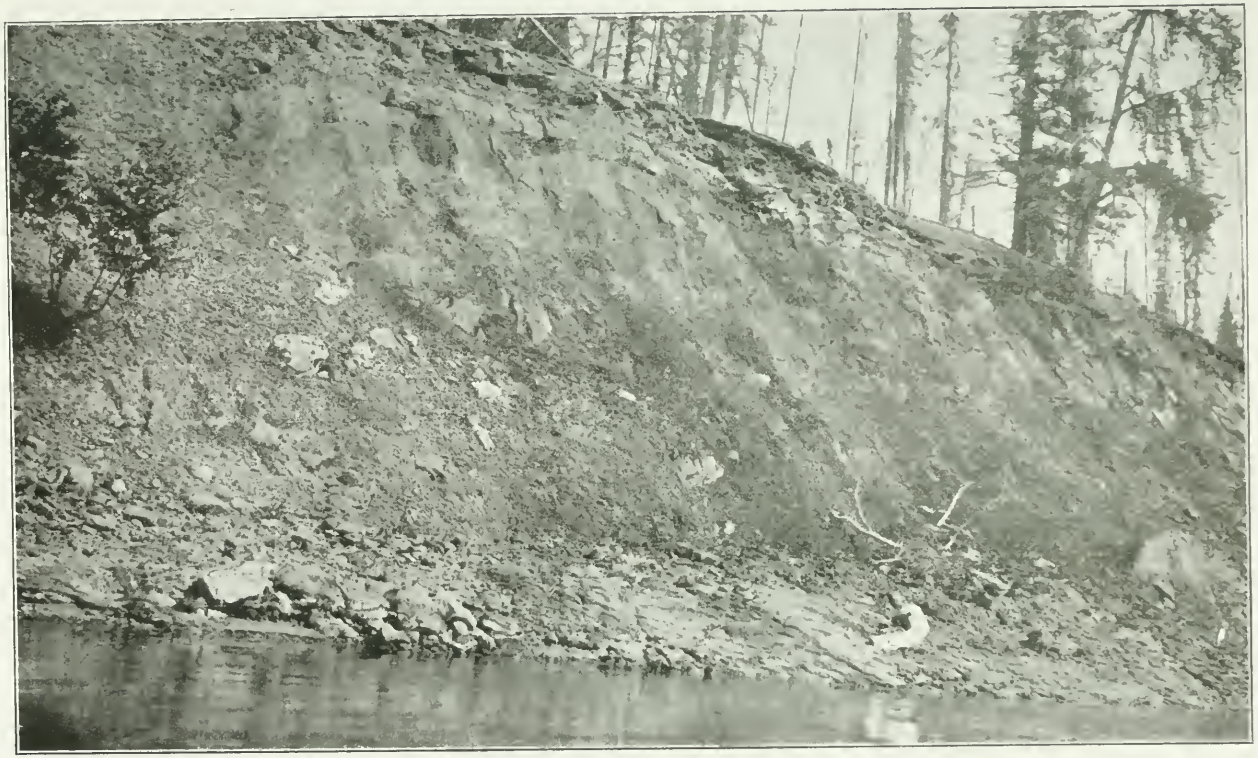
principle. From upwards of a dozen attempts at separation with which the writer is more or less familiar, the principal remaining evidence consists in a considerable number of disappointed investors and in the ruins of abandoned plants. Probably the two most ambitious attempts at commercial separation resulted in the construction of plants at Carpinteria and at Sisquoc in the State of California. The former is said to have represented an expenditure of approximately $300,000 and was abandoned prior to 1899. The latter, which was abandoned about 1901, is said to have represented a total financial outlay of approximately $1,000,000. Such results at the outset, adversely affected the exploitation of the new material.

The early history of the use of bituminous sand as a paving material, was also marked by many failures, but it is now recognized that in the majority of instances this was due chiefly to lack of appreciation of the true character of the material used. With correct understanding of the true character of the bituminous sand, and by intelligent manipulation, excellent pavements are now being constructed with this material in California and Oklahoma, and to a lesser extent in Utah and Kentucky.

Thus, at the outset, in considering the economic possibilities of our own Canadian deposits, the outstanding features referred to above regarding the commercial history of bituminous sands in the United States were not encouraging. Moreover, in the United States many of the deposits were very favourably situated with regard to rail or water transportation, and were also, in many instances, adjacent to markets of importance. In Alberta, on the other hand, the most southerly of the workable deposits is approximately 300 miles by rail from the nearest centre of population. Consequently in planning an investigation of the Alberta deposits, a knowledge of the history of the development of somewhat similar deposits in the United States was of very real value, and an attempt was made to profit by the costly experience of others.

There are three lines along which Alberta bituminous sand may possibly be developed commercially. These are:

1. Use of the bituminous sand in a more or less crude form as a wearing surface for streets and highways.



Exposure on east side of Athabaska river, 43 miles north of McMurray. In descending the river, the thickness of the overburden gradually decreases.

2. Various commercial applications of the bitumen when separated from the associated sand aggregate.

3. Destructive distillation of the crude bituminous sand with recovery of crude petroleum.

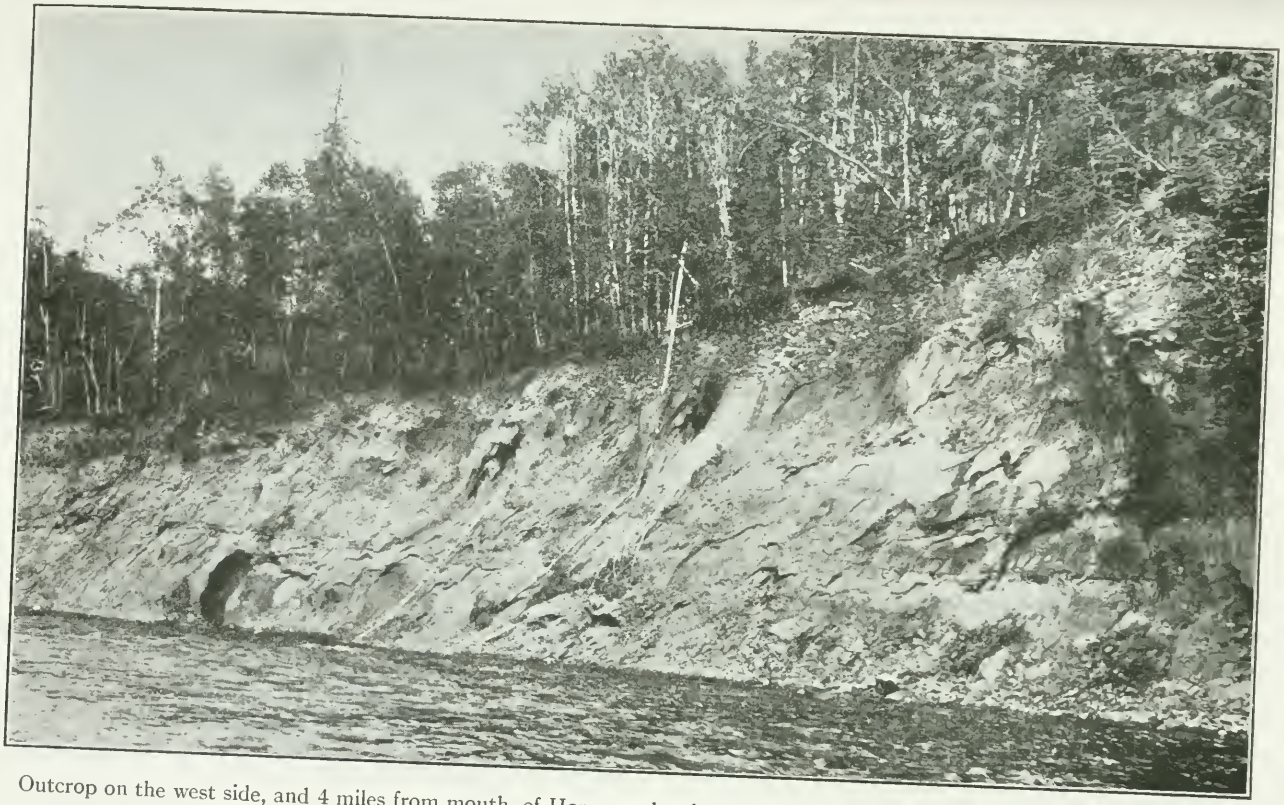
In the following outline of the present investigation each of these possibilities will be referred to briefly.

In 1913 a small field party was organized and during July and August a general reconnaissance of the McMurray area was made. This work included the measurement of a large number of outcrops and the collecting of numerous representative samples. Subsequently the writer made a careful laboratory study of these samples.

Results of the field and laboratory work of 1913 were sufficiently encouraging to warrant a more detailed study of the deposits. The question of the adaptability of the Alberta bituminous sand as a satisfactory paving material also required an answer. Certain well-informed engineers regarded the material as unsuited for the construction of asphaltic surfaces, partly on account of the high penetration (250+ Dow Standard) of the bitumen itself, and partly on account of the unbalanced grading of the associated sand aggregate. Moreover, the results of small scale laboratory investigation were inconclusive. The writer, therefore, suggested that as a practical test an experimental pavement should be constructed.

Considering the importance that would undoubtedly attach to the results of the proposed paving, great care was required in selecting the outcrop or outcrops from which a first shipment should be taken. It was already evident that wide variation existed in the grading of sand aggregate of various exposures, while the character of the bitumen itself also varied. Consequently upwards of sixty core samples were secured at various points over a large area and analyses for grading of sand and for percentage of bitumen were made in a field laboratory brought for that purpose. This boring involved a large amount of arduous work, not only in the actual boring operations, but in transporting up and down swift and shallow streams, drills and forge and the usual equipment of a fairly large field party.

Eventually it was found impossible to secure a satisfactory material from any single outcrop. It was, however, evident



Outcrop on the west side, and 4 miles from mouth, of Horse creek; showing massive bituminous sand under light overburden.

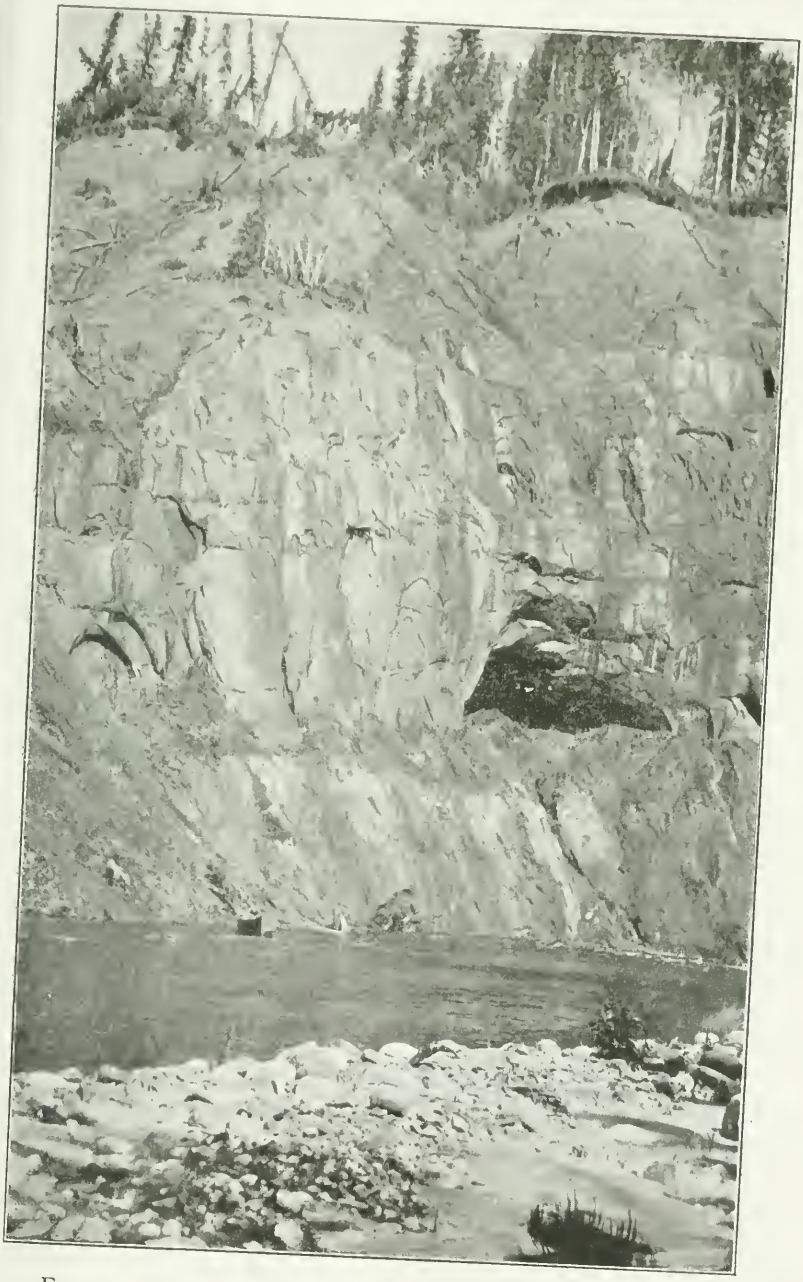
that by combining the bituminous sand from two separate outcrops, in proper proportion, a satisfactory product could be obtained. Two outcrops, therefore, were selected and a shipment mined and sacked.

The transportation of the material from points 40 miles beyond McMurray to Athabaska—upwards of 280 miles—involved considerable difficulty. Water transportation was out of the question, but during January and February, 1915, twenty-three teams made the trip down to McMurray and eventually delivered the shipment of bituminous sand at Athabaska. During the summer of 1915 this material was used by the writer in the surfacing of a portion of Kinnaird Street, Edmonton. Small areas of sheet asphalt, bitulithic and bituminous concrete were laid. It is unnecessary here to refer to the procedure and manipulation adopted, as an outline has already appeared in a publication issued by the Mines Branch. Recently Mr. A. W. Haddow, Acting City Engineer of the City of Edmonton, reported to the Department that the above pavement was in perfect condition.

The economic importance of bituminous sand in general when used in the construction of wearing surfaces may be very briefly stated as follows:

1. Natural bituminous sand or rock asphalt is a mechanical mixture of sand and asphalt or asphaltic petroleum, and it has frequently been found that there is very little uniformity in such material. There are, of course, notable exceptions—as for example at the quarries of the City Street Improvement Company, near Santa Cruz, Cal., where a large tonnage of crude bituminous sand is of an unusually uniform character. In the case of the Alberta material, detailed exploration by means of adequate equipment will be necessary before the degree of uniformity of any considerable tonnage will be known.

2. In the preparation of artificial mixtures, a more or less exact product may be had, the quality and percentages of sand and of asphalt cement being readily controlled. Thus a suitable mixture, once established, can be maintained uniformly. Assuming that in certain instances conditions will warrant the use of bituminous sand as a paving material, the necessary manipulation will present no serious difficulty.



Exposure on the north side of Steepbank river, about 3½ miles from its mouth, illustrating typical massive structure and cleavage common to high grade deposits of bituminous sand.

In California, torpedo mixers[1] for the treatment of bituminous sand have been perfected and are being successfully operated both as regards costs and actual mechanical operation.

3. In the case of many of the harder silicious rock asphalts found in the United States, serious difficulty has been met with in disintegrating the crude material. This difficulty would not be encountered in the manipulation of the comparatively soft Alberta bituminous sand.

4. Generally speaking, labour is the most costly element that enters into the construction of asphalt wearing surfaces. Sheet asphalt, bituminous concrete and bituminous macadam consist largely of filler, sand, and crushed rock, each of which may usually be obtained without difficulty and at low cost. Of the materials required other than sand and rock, the most costly is asphaltic cement, but as the percentage used is relatively small, its proportional cost is not great. In certain parts of the prairie provinces, however, suitable sand and crushed rock are difficult to obtain at a reasonable cost. Doubtless such a condition would favourably affect the demand for crude bituminous sand.

At various times during the past three years, the writer has discussed the commercial application of bituminous sand in the surfacing of streets and highways with municipal engineers in upwards of twenty cities and towns in Kentucky, Missouri, Oklahoma, California and Utah. Pavements in these States, surfaced with various classes of bituminous sand, have also been examined carefully. The conclusions arrived at have been briefly summarized in a publication already issued by the Mines Branch[2].

As noted above, experimental street paving in which Alberta bituminous sand was used for the first time on a commercial scale, was completed by the writer in Edmonton, Alberta, in 1915. This surfacing was laid on a six-inch concrete base, and after two years' service under fairly heavy traffic, still appears

---

[1]Manufactured by Schneider Engineering Works, San Francisco, Cal., Columbia Machine Works, San Francisco, Cal.

[2]Preliminary Report on the Bituminous Sands of Northern Alberta, appendix II: S. C. Ells.

to be in a satisfactory condition. It is clear, however, that in the extensive construction of interurban roads and highways in the western Provinces, the cost of such a concrete base would be prohibitive.

The present problem appears therefore to be no longer a study of the actual durability of the bituminous sand itself as a wearing surface. Rather does it appear that the extent to which this material may be used in the surfacing of rural roads and highways will depend on the following:

(*a*) Haulage charges on such a bulky low-grade material as crude bituminous sand will restrict its use within comparatively narrow limits. These limits would greatly be extended as a result of the development of a satisfactory process for the commercial separation of the bitumen from the crude bituminous sand.

(*b*) The development, if possible, of a comparatively inexpensive type of road construction in which the customary expensive concrete base may be eliminated. This appears to be essentially a problem for highway engineers who are thoroughly familiar with local climatic conditions and also with the soils of which the sub-grade would be composed. Obviously under existing conditions of soil and climate, the problem could be undertaken with better chances of success in the southern portions of the province of Alberta than in the northern portions.

During the past twenty years many statements have appeared regarding the extent of the McMurray deposit. Liberal estimates of area and of average thickness were usually accepted and the calculated tonnage was, of course, very large. In arriving at these conclusions, however, three important features have usually been overlooked. These are transportation, variation in quality of material, and overburden.

As regards transportation of a bulky low-grade material, it is obvious that a large part of the area underlaid by bituminous sand will, for a long period, remain quite inaccessible to development. Moreover, in many parts of the area, it has been found that the bituminous sand is of such a low grade, as to be of little or no value. Lastly, and assuming that all excavation must take the form of open cut work, the overburden becomes a factor of dominant importance. Even taking these three features

into consideration, however, there still remains commercially available a large tonnage of bituminous sand. In order to arrive at some definite conclusion regarding the probable tonnage actually available, and also in order to indicate those outcrops which were best adapted to commercial development, topographical maps of a large area drawn on a scale of 400′—1″ have been prepared by the writer.

In any attempt to ship crude bituminous sand, freight charges would seriously handicap a large commercial development of the McMurray deposit. This is, of course, obvious in considering a material of which 85% to 88% is sand and but 12% to 15% is bitumen. Consequently, the desirability of developing a successful commercial process whereby the bitumen might be shipped in a more or less pure form becomes apparent. Recently, therefore, an effort has been made to determine whether a successful separation process might not be developed.

In a preliminary investigation carried on by the writer at the Mellon Institute of Industrial Research at Pittsburg, Pa., some interesting results have been obtained.

At the outset it was necessary to choose between separation by means of solvents or by means of heated water. After two months of experimental work in the laboratory, and after carefully considering available data relating to previous separation work in various parts of the United States, it appeared that separation by means of hot water afforded more promising chances of success. Moreover, it was recognized that, within certain limits, the higher the temperature of the water, the more efficient was the resulting separation. Eventually experiments showed that the most efficient temperature for such treatment of Alberta bituminous sand lay between 315° and 330° F.

The adoption of such a temperature implied the use of apparatus designed to carry a pressure of 85 to 105 pounds. Serious difficulty was encountered in constructing this apparatus and the series of trial laboratory runs is not yet completed. It should be noted that the principle of using pressure apparatus for the treatment of bituminous sand has been patented in the United States by C. H. Cook and J. R. Price, both of San Francisco.

In considering the derivation of crude petroleum from crude bituminous sand, a quantity of crude oil was distilled from the bituminous sand. This oil was then completely fractionated and each fraction refined. Based on the data thus secured, and making the necessary assumptions regarding other factors, a tentative estimate was prepared showing investment, working costs, and probable gross profits from commercial destructive distillation of the crude bituminous sand.

The capacity of the retort used in carrying out this distillation in the laboratory was approximately fifteen pounds of of crude bituminous sand. It will obviously be necessary to extend the work on a much larger scale before anything definite can be stated regarding the commercial possibilities of such distillation.

NOTES ON ELECTRO-CHEMICAL AND METALLURGICAL POSSIBILITIES IN CANADA

By H. E. Howe

Annual Meeting, Montreal March, 1917

In considering the utility of hydro-electric power we must remember that it is simply another way of selling energy, and its cost must always be compared with the cost of energy in other forms, such as coal. It may be expected, therefore, that hydro-electric development will be more extensive in the immediate future in Canada than in for example the United States, where coal is found relatively near the industrial centres and at points where it can be used direct or where gas from by-product coke ovens can be used to produce electric energy at a price competing with its cost when generated by hydro-electric power. It must be borne in mind that while steam plants are now producing power with approximately one-half the coal required fifteen years ago and at a capital cost of little less than half what it was at that time, hydro-electric equipment had a very high initial efficiency and consequently has not enjoyed anything like the same degree of improvement, while the cost has in no way decreased. In fact, owing to increased prices of materials and labour, the cost per horse-power is probably higher than it was fifteen years ago.

Therefore, the fact that a water power exists and offers opportunity for development as a source of electric energy does not necessarily imply that it is economic to develop it or that developed, a group of industries will at once find a home in that vicinity.

Let us consider for a moment whether electro-metallurgical and electro-chemical industries may be established advantageously in Canada at the present time. To determine this, two points must be considered: First, is there sufficient demand for the products of these industries in Canada, and second, is there not other demands for electric power which at present must supersede such industries? Almost without exception electric products are the raw materials for other industries; we must have a relatively large number of such industries before the establish-

ment of electro industries can be justified. As to the second point, it would seem that a market for labour is one of the commanding questions before Canada at the present time, especially in view of the return of soldiers and the expected immigration of many hands who will want employment in other than agricultural pursuits.

Other industries, such as spinning and weaving, employ many more hands per horsepower of electric energy consumed than do electro-chemical and metallurgical plants and can afford to pay a higher price. Such employment in turn builds up communities and offers opportunities for wholesale and retail merchants and the host of secondary industries incident to any concentrated population. If we consider the relation between the horsepower consumed in Niagara Falls, N. Y., or Ontario, in relation to the population of these places and apply the same line of reasoning to the locality of Manchester, N.H., or Lowell, Mass., this point will at once be emphasized. It must be obvious that electro-chemical and electro-metallurgical industries face another handicap, for with the increase of the consuming population and industries utilizing electro products there comes a growing and insistent demand for power which tends to place the cost of electric energy at a higher price than such processes can afford to pay. They are therefore gradually forced to seek advantageous points at a little distance from the large centres, but these points can only be chosen where the reasonably low incoming and outgoing freight rates are available.

The considerations determining the location of electro industries, include the following: First, there must be a source of large amounts of power. The fixation of atmospheric nitrogen, for example, requires 25,000 kilowatt hours to produce one ton of 100% nitric acid. The manufacture of aluminum requires a like amount of power per ton of finished product. The production of ferro-alloys which have become an essential in modern industry for the making of high speed steels, the hardening of armour, and piercing projectiles, consume from 3000 to 8000 kilowatt hour per ton of product depending upon the grade of alloy made. The process of casting an alloy or refining steel consumes from 600 to 1000 kilowatt hours per ton, while the refining of nickel requires energy to the extent of 3000 kilowatt hours per ton. Lead is the

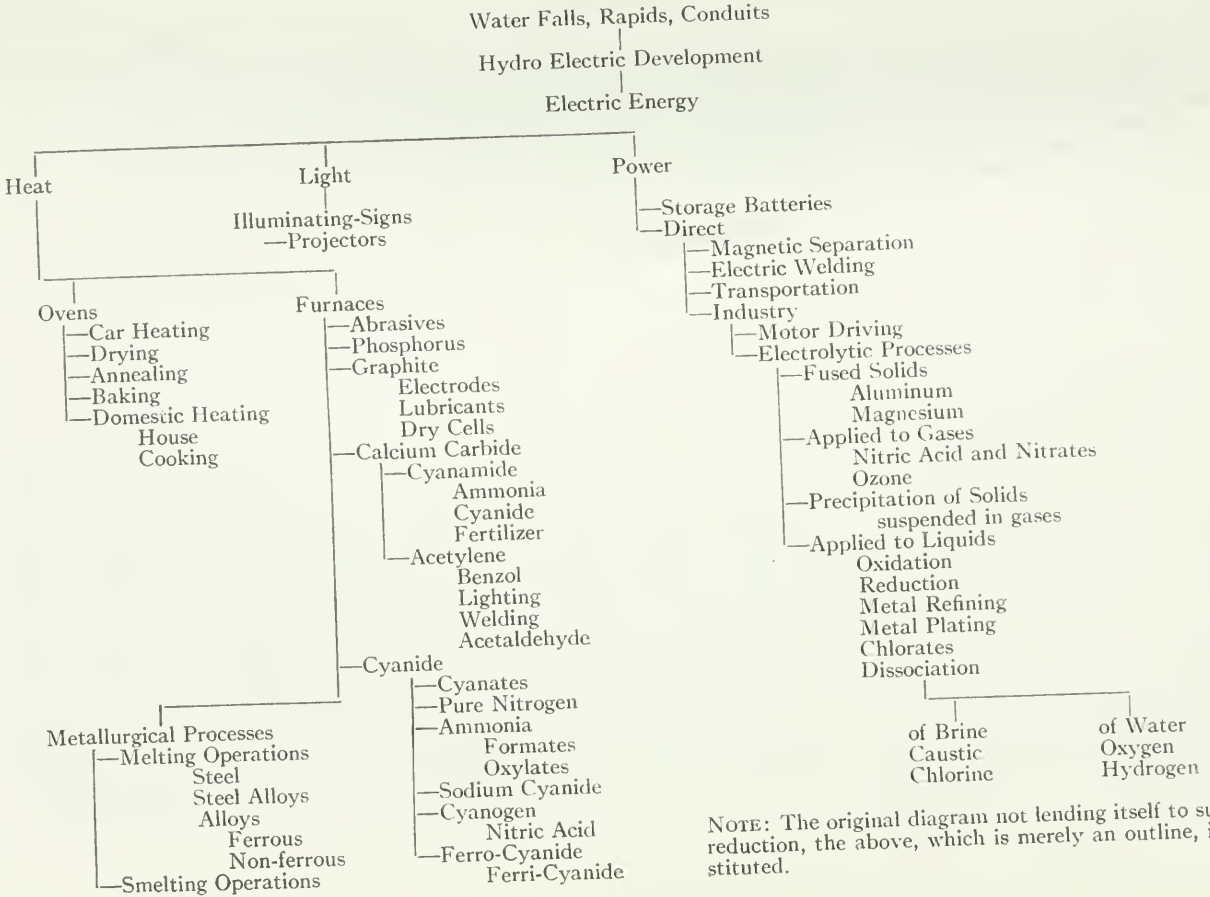
smallest power consumer, since only 120 kilowatt hours per ton of refined product are required, while copper at 300 kilowatt hours per ton is next.

Second, this power must be reliable. It should not be subject to great variation, for in electric furnace operations, especially, it is frequently fatal to have a sudden diminution in the amount of power available. This means that in some instances a water power which is subject to severe ice conditions or where the flow of water materially decreases in summer must have a steam plant as an auxillary if a high average of efficiency is to be derived from the water power itself during other seasons of the year.

Third, the power must be cheap, far cheaper than many suppose, if electro-chemical and metallurgical processes are to be established on anything approaching a large scale. $20.00 per horsepower year, which is equivalent to 3c. per kilowatt hour, is not cheap. At that price many important processes cannot be carried on. By locating a plant equipped with the modern turbo alternators near the coal fields, it is possible to compete with this figure through the generation of electricity from steam power. Artificial fertilizers and the great group of industries depending upon nitrates must have power considerably below $10.00 per horsepower year. Dr. Eyde stated in 1912 that when electric energy in Norway exceeded a price of $6.00 per horsepower year it would become impossible for the great plants, so favourably located, to compete with other sources of nitrates.

Fourth, there must be reasonably cheap freight rates from the plant to the consuming industries and on the raw materials which must be brought in. Electro-chemical industries are large consumers of such materials as salt, coal, silica and calcium compounds, and in many instances their products must be sold at a price which precludes the possibility of paying large amounts for freight.

Fifth, for this reason the nearness to consumers is almost always an essential. It has been estimated that a distance of 1000 miles to the centre of distribution is equivalent to an increase of from $10.00 to $20.00 per horsepower year in the cost of power



NOTE: The original diagram not lending itself to suitable reduction, the above, which is merely an outline, is substituted.

itself. Where raw materials are bulky, and the product is not bulky, work must be carried on near the raw materials. Where large amounts of power are essential it may pay to bring the raw materials to it. Such questions must be considered separately in each individual instance.

Sixth, there must be good labour conditions, and finally raw materials must be available.

Relatively higher prices can be paid for electric power only where the optimum conditions of locality, market, freight rates, accessibility to raw materials and labour conditions exist. An exceptional advantage for any one of these five factors may be sufficient to a overcome a considerable weakness in one or more of the others; for example, in Norway power is so cheap and a large market so near, with water transportation, that a very large carbide plant is being installed, even though some of the raw materials must eventually come from a considerable distance. Niagara Falls is a most unusual locality, and because of its exceptional advantages must not be taken as a criterion for possible developments in other places as they exist at present.

The diagram (p. 463) is intended to show the economic importance of electro products and their relation to other industries. The classifications are somewhat arbitrary and many more uses are found for the products than can be conveniently indicated in such a chart. While it shows the tremendous importance of the products of electric energy, it also indicates that in nearly every instance the final products are the raw materials of other industries. Few of them can afford to pay much for transportation. Two extremes may be considered. Aluminum, where tremendous power is required but where the finished product has a high value per ton and weight in proportion to its bulk that permits it to be transported great distances. We, therefore, find bauxite being imported from considerable distances into Canada. On the other hand, sodium hydroxide or caustic uses relatively small amounts of power and produces material not well suited to long hauls. The result is the location, where possible, of plants near the raw material-salt-and consuming centres, and it pays to transmit the power to such locations.

It would seem, therefore, that while Canada's hydro-electric

power offers many potential opportunities, it must be considered at present a reserve, and several conditions must be satisfactory before definite recommendations can be made. First, we must know about Canadian natural resources that lend themselves to the treatment of electric processes, and we must have information in more available form than at present; second, the selection and establishment of an industry utilizing electric energy must be done with great care and with a thorough study of the economics in each instance; third, there must be a larger consuming population in Canada before the introduction of many of the electric industries can be justified; fourth, there must be more industries using electro-products as their raw materials, and fifth, really cheap power must be avilable.

To bring about this latter condition there must be lower costs for development. Many of the great power companies have been obliged to spend considerable money in embellishment, owing to the location of their power plants, which has added much to the cost per horsepower. Owing to several uncertainties, money used in development has had to pay a higher interest rate than in other forms of industrial activity, and besides it costs nearly three times as much to develop a water power as it does to construct a steam plant to produce the same amount of kilowatts per hour. If capitalists would accept bonds at 5% instead of 7% the cost of production of a horsepower hour would be reduced 22%. Railways and other forms of industries seem to have no difficulty in obtaining money at this rate, and it may eventually become necessary for governments to use their credit to obtain money at low rates to assist in water power development. There must also be reasonable and co-operative legislation to insure the permanency and safety of private investments in such projects.

Based on these principles the immediate future seems to offer Canada two fields for the utilization of hydro-electric power in electro-metallurgy and electro-chemistry. Perhaps the most important is the electro-metallurgical field. Here Canada has the raw materials; she has the consuming market for much of the material to be used in the construction work which must go forward, while in many instances the products can be exported to advantage by all water routes. Hydro-electric power would seem particularly well adapted to this industry here, because

electric furnaces do not require the peculiar crucible clays which would have to be imported; the product is of the highest grade, and the fuel which we should feel called upon to conserve for the sake of posterity is replaceable with electric energy in the form of heat. Every million kilowatts used is equal to eleven and a half million tons of coal per annum, and it must be remembered that under general average conditions it requires approximately four pounds of coal to develop one horsepower. Electro-metallurgic processes have already a strong foot-hold in Canada and important developments have been the result. I refer to the production of ferro-titanium directly from the ores and concurrently with it the manufacture of high-grade irons from the titaniferrous iron ore of Canada, which have constituted a problem. You are also aware of the important place which electro-metallurgy holds with relation to munition manufacture. The indications therefore seem to be that further development, not only in the smelting of ores but in the utilization of scrap materials lies in the immediate future.

In the electro-chemical field there are also instances of important developments, such as the group of processes at Shawinigan and at Niagara Falls, Ontario. The production of nitrates now illustrated by the American Cyanamid Company at Niagara Falls, Ontario, would seem to offer the best electro-chemical field, but it must have far cheaper power than is now offered if considerable expansion is to be expected. The production of nitrogen fixation is of great economic and national importance. Nitric acid is the basis of many chemical industries, is an absolute essential in all explosive manufacture, and is required in ever-increasing amounts for the production of fertilizers. Canada must eventually use fertilizer. It is well known that we cannot continually draw money from the bank without occasionally depositing funds, and yet we hear stories of how rich the farm lands used to be forgetting that the fertility of a virgin soil must gradually disappear unless replaced in some measure. Those nations praised for the yields per acre are also the nations where the greatest amounts of fertilizer are used annually per acre. The fixation of atmospheric nitrogen is a large consumer of power, the product will stand reasonable transportation charges, a nation cannot be wholly independent without nitrates, and for these

reasons this particular electro-chemical process would seem to command early attention.

Although it is beyond the province of this discussion, the utilization of hydro-electric power to a greater extent in transportation may be mentioned. The heavy initial investment is a serious factor, but against this may be mentioned the conservation of fuel, the fact that many water powers exist where grades are heavy and population sparse, and the fact that considerably greater motive power can be obtained from an electric locomotive than it is possible to develop from the most modern steam locomotives.

In considering the possibility of saving coal through the utilization of hydro-electric power, it is interesting to note that prior to the war France consumed about 60,000,000 tons of coal per year, of which 33⅓ % was imported. Now the Germans hold a good part of the French mines, and it is needless to say that when they are returned to France they will be well worked out. Consequently a survey of the water power resources of France has been made with the result that power estimated at from 6,000,000 to 10,000,000 horsepower has been located. About 45,000,000 tons of coal are used annually in France for power producing 12,000,000 horsepower. The development of 6,000,000 hydro-electric horsepower will represent an annual saving of over 20,000,000 tons of coal.

The cheapness of electric power determines to a great degree the future of electro-chemistry and electro-metallurgy in Canada, as well as elsewhere. I have attempted to set forth some of the limitations governing present processes, but with cheaper power it is not at all unlikely that new processes wil be devised and new products be produced that may entirely alter the situation as it appears at present. Cheap power will also largely determine what can be done with existing processes.

Discussion

Dr. W. L. Goodwin—I have travelled in Ontario a great deal and I believe we could develop any amount of water power, if it could be supplied cheaply enough. The paper just read re-

fers to that point. I remember some years ago when I prepared a paper for the Institute on the general subject of water-power, I noted that water-power in Europe was then being sold for, I think, $7 per h.p. per year. Mr. Howe states that a rate of $24 is too expensive, but I believe power can be bought in Canada for very much less than that figure—probably in many centres it could be supplied for $10 per h.p. per year.

Mr. W. J. Dick:—The efficiency and adaptability of electric energy for refining of metals and the manufacture of chemical products have long been recognized but it was not until recent years that the industrial world awakened to the importance of the new field. Electric refining, at first applied to copper only, is now being extended to all the metals and the electric current is also employed in their extraction from the ores. The manufacture of ferro-alloys of chromium, vanadium, molybdenum, tungsten, titanium, silicon, etc., required for special steels, has assumed an importance which is rapidly increasing. The production of aluminium, of calcium carbide, of the abrasives, of new refractory materials, of graphite, etc., has already created large industries. Sodium compounds and other well-known chemicals have long been manufactured by electrolysis. The fixation of nitrogen with its many subsidiary industries, such as the manufacture of nitric acid, ammonium nitrate, explosives, etc., the reduction of magnesium and the production of innumerable chemical compounds known at present only to the special trades requiring them, are now under commercial development.

On account of the low efficiency of electro-chemical work and the large amount of power used in the processes it might be said that one of the principal controlling factors in the establishment of these industries is cheap electric power. The water-power resources of Canada are not only great but there are many large water powers situated near tide-water which would be developed and operated at low cost. In addition, their situation and proximity to markets either on the Atlantic or Pacific offer unrivalled opportunities for the expansion of present chemical industries and the establishment of new ones. Owing to the advantages outlined above, and to the fact that electric current

in eastern United States will become more valuable for power and lighting purposes, Canada is destined to secure the principal electro-chemical and electro-metallurgical industries of eastern North America.

Norway has long forseen the importance of developing her water powers and today her electro-chemical products form the basis of many of the chemical industries in Europe.

The principal electro-chemical industries already established in Canada are:

Manufactures of aluminium, calcium carbide, metallic magnesium, calcium cyanimide, phosphorus, ferro-silicon, and the refining of copper, zinc and cobalt. The first five industries enumerated not only produce considerable quantities for home consumption but also for export.

In 1913, the United Kingdom imported some $23,250,000 worth of chemical products of which most of them could be manufactured as electro-chemicals but of this amount over 2,$750,000 worth were electro-chemicals such as ferro-silicon, spiegeleisen, calcium carbide, and some soda compounds. Of the total imports of chemical products Canada did not export more than $200,000 worth and of the electro-chemicals mentioned not a dollar's worth was exported to Great Britain. On account of the large powers situated near our shipping routes Canada has a splendid opportunity to secure this trade but instead of developing our own electro-industries we are developing our water powers, transmitting the electricity across the line and building up electro-chemical industries there. This policy is an unwise one for two reasons: first, we are developing some of our best water powers and allowing industries to be created in the United States based upon this power, although experience has shown us that one of the great drawbacks to the economic manufacture of mineral products in Canada is the fact that we would have to compete with well established plants to the south of the line. In other words we encourage their establishment for the sake of giving employment to a few power-house men, and then deplore the fact that we cannot compete with these plants. Second: if, owing to there not being sufficient demand for power in Canada, we allow it to be exported to United States, which results in the establish-

ment of vested interests in that country, it may be a difficult matter to prevent the export of this power when there is the demand for it in Canada.

The following extract from the Seventh Annual Report of the Commission of Conservation[1] shows the importance of this subject: "When development commenced at Niagara Falls, there was comparatively little market for electric power on the United States side of the river. Consequently, electro-chemical industries were induced to start, and large blocks of power were contracted for at a comparatively low figure. Considerable quantities of power were also exported from Canada to the United States, some of it being used in the newly started industries. The United States interests now fear that if they do not get the balance of the power that is already available, and can be made available under the existing treaty, at Niagara Falls, that the power will be used in Canada to build up Canadian industries. Before long, the electro-chemical industries, such as those manufacturing nitrogenous products, will be re-establishing themselves elsewhere. Under existing contracts they are getting at $10, $12 and $15 per horse power, power that has a market value in cities in New York state, ranging from $60 to $75 and upwards per horse power. Now, if these industries in the United States can continue to increase their importation of electricity from Canada for a little while longer, and if the available surplus from Canada can once be transferred to the United States, then, according to the New York Public Service Commission, there need be no fear of its being withdrawn. This is a matter of great importance to Canada."

[1]Water and Water-power Problems, by A. V. White.

MAGNESIUM

Its Properties, Manufacture and Uses

By J. C. King

Introductory.—Before the war magnesium was used in moderate amounts and was produced in Germany, which possesses large deposits of magnesium salts from which the metal can conveniently be produced. Magnesium has been needed in large quantities for military operations, and one effect of the war has been the establishment in Canada of a plant for the production of this metal after the German supply had been cut off.

Although the process for making magnesium was known in outline, a large amount of research, both in the laboratory and in the works, was required before the metal could be produced commercially in this country, and even now the process has not reached its full development.

The economical production of magnesium requires not only an amply supply of the raw materials and of electric power, but also depends on a large and continuous demand for the product. Military requirements, although adequate for a time, cannot permanently be depended on, and it becomes necessary to investigate the possible uses of this metal in alloys and for other purposes. The information available with regard to the properties, manufacture and uses of this comparatively new metal have been summarized in the following paper by Mr. King.—Alfred Stansfield.

Properties of Magnesium

1. *Physical:* Pure magnesium is a white, brilliant metal having an atomic weight of 24.32, and a specific gravity of 1.74; it melts at 651°C., and boils at 1120°C. Specific heat 0.249; lin. coeff. of expansion .000,026,9; thermal conductivity 0.376. C. G. S. units. Heat of combustion to MgO 6077 calories per grain or 10,939 B.T.U. per lb.; electrical conductivity 230,000 resistivity at 20°C. (michrohm-cm.) 4.569; temp. coeff. of resist-

ance 0.004,19; tensile strength 33,000 lb. per sq. in.; compressive strength 38,000 lbs. per sq. in., and bending strength 25,000 lb. per sq. in.

Brittle when cold, but extremely malleable and ductile when heated to about 400°C, magnesium alloys readily with most metals and greatly modifies their crystalline and other physical properties.

2. *Chemical:* Magnesium oxidizes readily when exposed to the atmosphere, forming a thin coating of hydriated oxide which prevents further corrosion. In a finely divided state magnesium may be ignited, when it will burn in the air, and, if mixed with an oxidizing agent, combustion is produced with explosive violence, accompanied by the emission of great heat and a dazzling white light rich in actinic qualities. Magnesium is a most valuable reducing agent, due to its intense avidity for both oxygen and nitrogen.

Pure magnesium has no action on pure water, but in the presence of a trace of platinum chloride it decomposes water quickly, evolving hydrogen. Platinized magnesium constitutes a good reducing agent, for instance, it reduces nitrobenzine completely to aniline (Ballo. Ber. 16–694). Dilute acids rapidly dissolve it with violent evolution of hydrogen. Strong sulphuric acid acts slowly and a mixture of this acid with fuming nitric acid has no action on magnesium at ordinary temperatures. Solutions of caustic alkalis have no action upon it. Magnesium combines with sulphur, phosphorus and arsenic, and at a red heat decomposes oxides of carbon, SO_2 and many hydro-carbons. Silica and boric acid are readily reduced by it, and it combines directly with nitrogen, forming a crystalline nitride of the composition Mg_3N_2. Magnesium precipitates nearly all metals from their neutral solutions in the metallic state, but aluminum uranium and chromium are precipitated as oxides.

MANUFACTURE

Metallic magnesium has been produced in Germany by electrolysis from a fused dehydrated bath of natural carnalite, *i.e.*, a double chloride of potassium and magnesium. The metal

produced had a purity of from 96% to 98%. The Shawinigan Electro Metals Company are the only producers of magnesium in Canada. They are using a process which consists in electrolyzing a bath of fused chlorides and their process is such that they are able to produce a metal of a purity of 99.7% magnesium. This has been accomplished through improvements in furnace design and modifications in the composition of the electrolyte, etc.

The chlorine is collected and used in the manufacture of chloride of lime and magnesium chloride ($MgCl_2$). Magnesite ($MgCO_3$) is calcined and ground to a very fine powder which consists of magnesium oxide (MgO), with impurities such as iron, alumina and silica. Water is added to form milk of magnesia (Mg(OH)2), which may be treated in either of two ways to obtain magnesium chloride: (1) Chlorine may be passed through milk of magnesia in the presence of a catalizing agent, or (2) chlorine may be converted into hydrochloric acid gas (HCl) which is passed through the milk of magnesia. The dilute liquor obtained by either of the above methods is then purified and evaporated to dryness, the magnesium chloride being returned to the furnaces. This process, it will be seen, is cyclic with respect to chlorine.

Uses

Up to the present, magnesium has been useful, mainly through its property, when finely divided, of rapid combustion in the air, with attendant emission of heat and light, as in flash light, flare, shell trailer and star-bomb. These uses, while important, are but temporary. It is to the metallurgist, not to the munition manufacturer, we must look for the development on permanent lines of the magnesium industry.

Prof. J. W. Richards, of Leheigh University, says: "While the war lasts, with its enormous demand for magnesium for military purposes, the price will remain in the dollars per pound, but with improved processes, and with normal industrial conditions after the war, magnesium will probably sell at a price which will take it out of the class of rarer metals, and put it among the common ones. As the price goes down its industrial uses will increase in geometrical proportion, and instead of the production being expressed in thousands of pounds per year, it will

reach thousands of tons. This will be one of the by-products of the war's stimulus to metallurgical industry. The possibilities held out to the metal industry by reasonably cheap magnesium are extremely interesting, and it is quite possible that alloys analogous to dur-alumin may be discovered as strong as soft steel with only 30% of its weight, which will find extensive use in aeroplanes and dirigibles. Such alloys may largely displace aluminum alloys, which are now used by the thousands of tons annually in the automobile industry, with a saving of one-third in weight, which will compensate for the higher first cost. Other metallurgical uses of magnesium which will be greatly extended by its lower price, are for deoxidizing brass, copper, bronze, nickel and monel metal, since it is a much stronger deoxidizer than aluminum. Aluminum has, in fact, blazed the way into numerous uses for which magnesium, as soon as it becomes cheaper, will compete and replace its older sister."

Dr. Wm. H. Grosvenor says: "The chief uses of magnesium are as a scavenging alloy, *i.e.*, for clearing up oxides of other metal, and making for denser, cleaner, stronger and more homogeneous alloys. In aluminum casting, for instance, magnesium cleans up the aluminum, about doubling its tensile strength, and quadrupling its resistance to shock or jar. It reduces the cost of machining—halving the number of resets on the cutting tool, and giving clean cut machine surfaces. The increase in strength means a reduction in weight of casting of generally not less than 25%. A number of experiments have been made on deoxidizing copper, and these show that copper containing oxygen can always be refined by means of magnesium. The copper used was electrolytic of great purity (99.96%) being free from tin, iron, lead, arsenic and sulphur, and containing 0.01% zinc, a trace of nickel, and 0.0237% oxygen, or a total impurity of less than 0.04%. When the melting took place rapidly under a charcoal cover, a few hundredths of a per cent of magnesium was sufficient, the required amount varying with the speed of melting and also with the length of time for which the melted copper remained in the furnace after the melt was over. An alloy of magnesium and copper containing 20% magnesium was preferred for these experiments to a richer alloy, as its use prevented a too violent reaction and loss in manipulation. This alloy is brittle and may

conveniently be weighed. An acid slag of the composition $FeO\text{-}CaO\text{-}2SiO_2$, finely powdered, and easily fusible, was added after the deoxidization had taken place and proved advantageous, until much oxide had been taken up, when the slag was made more liquid by the addition of building sand.

"Special attention is necessary to see that as little of the refining (reducing) agent as possible remains in the copper after it has been used for deoxidizing. If to melted copper additions are made of other more easily oxidizable elements like tin, zinc, nickel, iron, manganese, phosphorus, silicon, aluminum or magnesium, the whole of the cuprous oxide (Cu_2O) present is theoretically reduced with formation of oxide of the other metal. However, on account of the great dilution in the solvent (copper in this case) an excess of the reducing agent is necessary. *This excess must be the greater, the smaller the tendency of the reducing agent to form oxides.* If this excess remains in the copper it may prove deleterious. In the series of metals above, the affinity for oxygen increases from first to last mentioned. Magnesium is the last, and has the strongest affinity for oxygen, in consequence of which, its excess in the solvent is least, and the electric conductivity of the copper is actually increased, due to its perfectly scavenged condition.

"These experiments show that of all reducing agents magnesium is the most suitable in removing the last trace of oxygen from desulphurized, poled copper, or from melted electrolytic copper, the addition of a few hundredths of one per cent of magnesium being sufficient to produce the best quality of copper, especially that of the highest electric conductivity.

Material Tested	Elec. conductivity in % that of purest copper	Breaking strength lbs. per sq. inch	Elongation %	Number of twists radius 100 m.m.
Electrolytic copper 99.96% ..	97.1	60,850	2.0	5.5
Copper + 0.1% Magnesium, 0.013%Mg. remaining in Cu	98.5	61,000	2.75	8.0

REMARKS: Starting material, total impurities... . . .04%
Total impurities remaining. 03%

"Magnesium may be advantageously employed in place of zinc in toxicological investigations[1], owing to its freedom from arsenic. It may likewise be used instead of zinc in volatic batteries owing to its higher electromotive force."

ALLOYS

Magnesium alloys with nearly all metals. It forms malleable alloys with potassium and sodium which decompose water at ordinary temperatures. Hoffman and Suchy claim that magnesium and its alloys may be cast free from hydrogen bubbles and without any oxide or nitride being formed, when mixed with from 0.1% to 0.5% of calcium. The calcium may be applied as powdered burned lime, and being used in small amounts it does not impart to the alloy any of the characteristics of a calcium alloy. It has been found, in the production of alloys containing a relatively low percentage of magnesium, that the magnesium is most readily and economically added in the form of rich alloys, which in general, when containing from 20% to 80% of magnesium, are quite brittle and can easily be weighed.

Aluminum-Magnesium: Perhaps the best known alloy of these metals is magnalium, which contains from 2% to 20% magnesium and from 80% to 98% of aluminum according to various authorities. It has a specific gravity of 2.4–2.6 and its properties are sufficiently useful to insure it a prosperous future. It does not smear under the file, has a fine grain, may be turned in the lathe even at high speed, and may be polished like a mirror. It is easily melted and cast, is used for many parts of machinery, for cooking utensils and scientific apparatus, and as it retains its silvery lustre very well, is suitable for optical mirrors, etc.[2]

Another alloy produced by Pistor and Rakowics of the Chemische Fabrik Griesheim Elektron, containing 92% magnesium and 8% aluminum has a specific gravity of only 1.75 and "strength and extensibility equal to the best gun metal." These metallurgists also state that alloys of magnesium and zinc, or of magnesium and copper are "greatly superior in their physical properties to the usual aluminum or its alloys." They propose the

[1]Roussin, J. Pharm. Chem. 3,413.
[2]Molinari.

use of these "high percentage magnesium alloys containing from 80% to 99.5% magnesium and from 0.5% to 20% of other metals for any mechanical purposes, especially automobiles, airships, instruments and for military and sporting purposes. The Shawinigan Electro Metals Co. has produced an alloy containing 50% magnesium and 50% aluminum. This is a rich alloy intended for use in producing low magnesium alloys of aluminum. It is silvery white, does not tarnish on exposure to air, and is extremely brittle, being readily crumbled in the hand.

Copper-Magnesium: Alloys of these two metals containing from 10% to 80% of magnesium are brittle. An alloy containing 10% magnesium and 90% copper produced by the Shawinigan Electro Metals Co., was about as brittle as cast iron, the surface of cleavage showing a crystallic structure and iridescent light orange colour. A 50–50 magnesium-copper alloy was also produced. This alloy is extremely brittle, having a crystalline structure and a steel gray lustre. It has the property of decomposing water at ordinary temperatures with evolution of hydrogen gas.

Chas. Vickers, of Niagara Falls, has patented an alloy containing magnesium 2%, titanium 5%, and copper 93%. It is believed that the titanium combines with the oxygen and other undesirable elements forming titanic acid, and the magnesium combines with them to form slags so fusible that they are entirely eliminated from the bath. The surface of the resulting castings are substantially free from undesirable specks and blemishes.

Zinc-Magnesium: Walter Rubel, Berlin, has patented an alloy containing from 94% to 99% magnesium, and from 1% to 6% zinc. It has a specific gravity of 1.8; an elongation of from 8% to 12%; tensile strength, when cast, 22,000 to 36,000 lb. per sq. in., when rolled, 36,000 to 50,000 lb. per sq. in. Satisfactory alloys of these metals cannot be produced with less than 1% nor more than 6% zinc. In case the material is to be rolled or pressed the zinc content should preferably not exceed 3%. Pure zinc is not essential, as good alloys can be made if the zinc contains copper, aluminum or metals of the iron group, such as iron, nickel, chromium or manganese. The alloy is prepared by carefully melting magnesium with the exclusion of air, and adding

zinc, when a readily fusible material results, which can be cast, wrought, rolled or machined like brass. The alloy presents the fracture of fine steel[3].

Lead-Cadmium-Magnesium: An anti-friction alloy of the composition 0.1% to 5% magnesium, 10% cadmium, and the balance (85% to 90%) lead, has been patented by E. A. Touceda. The magnesium is intended to harden and toughen both the lead and cadmium constituents.

Tin-Magnesium: An alloy of the composition 15% magnesium and 85% tin forms a brittle, lavender-coloured alloy, which decomposes water at ordinary temperatures.

Thallium-Magnesium: An alloy of these metals containing 5% of thallium has been found to be stable and much more malleable than pure magnesium[4].

CONCLUSION

From the foregoing it may be seen that while some interesting and important work has been done with a view of placing the magnesium industry upon a permanent basis, the surface of this field has been scarcely more than scratched. Much research into the uses of magnesium as a scavenging or alloying material has yet to be accomplished. In this connection it is suggested that the scavenging with magnesium, of aluminum, copper, brass, bronze, nickel and monel metal, might profitably be investigated.

The following tables of known and possible alloys of magnesium show what a wide field for research lies in this direction and may prove helpful to those working with such alloys.

---

[3]Met. & Chem., Eng

[4]Thorpe.

ALLOYS CONTAINING MAGNESIUM

Approximate Composition Per Cent.

	Mg.	Al	Cu.	Sn.	Zn.	Pb.	Various	Remarks
1	2–20	80–98	..	..	..	..	..	Magnalium Sp. Gr. 2.4–2.6 Pistor & Rakowics, Sp. Gr. 1.75
2	92	8	..	..	..	..	..	"Strength and Extensibility of Gun Metal," 32–38, 000 lb. per sq. in.
3	50	50	..	..	..	..	..	S.E.M. Co. Extremely brittle Sp. Gr. 22.
4	50	..	50	..	..	..	..	S.E.M. Co. Extremely brittle. Decomposes water at ord. temps.
5	10	..	90	..	..	..	..	S.E.M. Co. Brittle as case iron.
6	94–99	..	..	..	1–6	..	..	W. Rubel. Sp. Gr. 1.8. Tensile strength: Cast 22–36,000 lb. per sq. in. Rolled 36–50,000 lbs. per sq. in.
7	15	..	..	85	..	..	..	Brittle—lavender colour. Decomposes water at ord. temps.
8	95	..	..	..	..	..	Thallium 5	Very malleable.
9	2	..	93	..	..	..	Titanium 5	Chas. Vickers—pat. for castings.
10	0.1–5	..	..	..	..	85–90	Cadmium 10	E. A. Touceda—pat. Anti-friction bearing metal.

SUGGESTIONS AS TO POSSIBLE ALLOYS CONTAINING MAGNESIUM

Approximate Composition per cent.

	Mg.	Al.	Cu.	Ni.	Sn.	Zn.	Pb.	Various		Remarks
1	2–20	75–97	..	..	..	..	..	Thallium 1–5	Titanium 1–5	Modified magnalium.
2	2–20	75–95	..	..	..	..	..	Iron 1–3	Manganese 0.1–1	Modified magnalium.
3	2–15	75–90	..	3–10	..	..	..	Iron 1–3	..	Modified magnalium more non-corrosive.
4	1–15	85–90	5–12	..	..	..	..	Calcium 0.1–0.5	Titanium 0.5–3	Strong castings—light—hard
5	1–5	80–85	..	..	..	10–15	..	Calcium 0.1–0.5	Iron 0.5–1	Auto castings.
6	1–5	75–90	..	10–15	..	..	..	Manganese 0.1–0.5	Iron 0.1–2	Auto castings, stronger, heavier than No. 5.
7	5	85	5	..	5	..	..	..	..	Wire and sheet-high tensile strength.
8	1–5	10–15	80–90	..	..	..	..	Vanadium 0.1–0.3	..	Strong castings—heavy—hard.
9	1–5	10–15	40–45	40–45	..	..	..	..	Manganese 0.1–0.5	Modification of No. 8—stronger.
10	1–5	..	70–75	..	5–10	..	15	Phosphorus 0.1–0.5	..	Mg.—Phosphor bronze—heavy bearings.
11	1–5	75–85	..	..	5–10	..	10–15	..	..	Bearing metal—light—medium soft.
12	1–3	..	50–55	..	1–2	35–40	..	Manganese 1–2	Iron 1–2	Manganese bronze—high tensile strength.
13	trace	..	99.6–99.9	..	..	..	..	..	..	Purest electrolytic copper perfectly deoxidized.
14	0.5–3	..	..	..	..	..	80–90	Antimony 10–20	..	Modified type metal.
15	0.5–3	..	1–2	..	40–50	..	35–40	..	Antimony 10–15	Modified Babbitt metal.
16	1–5	..	26	70	..	..	..	Iron 1.5	..	Modified Monel metal.

SOME SUGGESTIONS ON FILING AND INDEXING

By Frank E. Lathe

Western Branch, 1917

A prominent metallurgist remarked to me some time ago that he had recently read a valuable article in one of the technical magazines, but was unable to recall when or where it had appeared. He added that he read practically everything that was written in English along the lines in which he was interested, but that much information was lost to him through lack of some good system of filing and indexing. He was but one of a very large class of engineers who feel they are too busy to spend the time necessary for any system of filing. There seems to be a general fear of adopting a system which will prove a white elephant, a system, in fact, requiring so much time to keep up to date as to leave very little for the actual reading and study of the subject matter. It is in the belief that there is between the two extremes a happy medium which it is entirely practicable for any engineer to adopt for his own use that this paper is presented.

It is not only in the preservation of technical literature that filing and indexing are useful to the engineer; it will also be found worth while to index mechanical drawings, records of assays, in fact, almost everything to which occasional reference is made. Even where a card index may not appear justified it will in many cases pay to secure an ordinary index account book, in which an alphabetical list may be kept of the papers to be referred to.

In this connection the following articles dealing with the subject of indexing may be consulted to advantage.: "An Extension of the Dewey System of Classification Applied to Metallurgy, Metallography and Assaying," by Robert M. Keeney, in the April, 1911, *Quarterly*, of the Colorado School of Mines. "An Extension of the Dewey System of Classification Applied to Mining," by Carl A. Allen, in the October, 1912, *Quarterly* of the Colorado School of Mines. "Bulletin No. 9, revised 1912," Engineering Experiment Station, Urbana, Ill.,

in which nearly all the engineering branches are extended as in the Dewey system. "Indexing and Filing Technical Literature," by Alvin R. Kenner, *Engineering and Mining Journal*, vol. 99, p. 851. "Filing Technical Literature," by Edward B. Durham, *Engineering and Mining Journal*, vol. 101, p. 784.

The classification generally adopted in libraries is the Dewey or decimal system. In it all literature is placed in ten main groups, each bearing a number, as follows:

0. General Works.	4. Philology.	7. Fine Arts.
1. Philosophy.	5. Natural Science.	8. Literature.
2. Religion.	6. Useful Arts.	9. History.
3. Sociology.		

Each of the divisions of useful arts bears the above number 6, and an additional numeral, thus:—

61. Medicine.	64. Domestic Economy.	67. Manufactures.
62. Engineering.	65. Communication and	68. Mechanic Trades.
63. Agriculture.	Commerce.	69. Building.
	66. Chemical Technology.	

Again, each of the subdivisions of engineering bears the above number, 62, with another distinguishing numeral, of which the second, or 622, stands for mining engineering. Similarly chemical technology, 66, includes metallurgy, which bears the number 669. This subdivision may be carried on indefinitely. Thus the successive numerals in 669.337.531 stand respectively for useful arts; chemical technology; metallurgy, metallography and assaying; copper; blast furnace copper smelting; the pyritic blast furnace and accessories; disposal of the gases; flue system. The periods are placed every three figures merely for convenience in reading.

Mr. Durham, in the article mentioned above, advises the use of the Dewey system, with some slight modifications. He confesses, however, that his file has outgrown shelving that occupies one whole end of his study, and has become a burden, his conclusion being that a filing system is something to be avoided by the engineer, unless he be a fixture. Mr. Kenner criticises the Dewey system, though in his own he has used the Dewey method while adopting independent numbering.

For libraries the Dewey system is probably as satisfactory as can be hoped for, and for the sake of uniformity if for no other reason it will doubtless continue to be used widely. It does not necessarily follow, however, that in its detailed extension it is the system best adapted to the use of the average engineer, even as one would hardly recommend the construction of an elaborate mill for the treatment of a few hundred tons of ore. For instance, it has too many diversions, and it will be found far more convenient to adopt a small number of main groups, and not attempt the subdivision of these to any considerable extent. Thus it will be desirable to have one place for flue systems, instead of placing them under the headings of copper blast furnaces, pyritic furnaces and reverberatory furnaces, as in the extended system. It will also be useful in many cases to substitute, for the strict group classification of the Dewey system, one in which recent articles will be most easily accessible, as they are the ones most frequently required for reference. It often happens, also, that one wishes to make more than ten divisions of a subject, and this is not provided for in the decimal system except by a subdivision of one of the main groups. It may be done readily, however, by substituting the letters of the alphabet for the numerals, when 26 divisions are available. By a combination of the two ideas this may be extended to any desired length; thus the main groups may be designed by numbers, 1 to 100 or more, while the divisions of these will be by letter, and the proper place in each division by number. In such a plan the number 24B71 would refer to group 24, division B and No. 71 in that division. Beyond this limit very few engineers will ever find it necessary to go.

All the above applies particularly to the filing of clippings from the technical magazines, except in the case of libraries, when it includes the arrangement of books. No filing system, however, will in itself fully meet the needs of the engineer. In connection with it a card index should be kept, this index being made as simple as possible but covering the transactions of the mining institutes and other books of a similar nature as well as the files. The index will greatly be shortened if the articles filed according to subject be not indexed at all. Then in looking up information one will first go over the division devoted to the

subject in question, and if more is desired consult the card index covering bound volumes and such articles as have necessarily been filed out of their own groups. Thus it will be possible in a few moments to collect all the available literature on any desired subject.

A few illustrations will now be given of the practical application of these modifications of the Dewey system to various conditions. But it should be remembered that these are only intended to be illustrative, and one engineer need no more think another's filing system will exactly fit his own needs than expect to wear with comfort the latter's hat, coat, and boots. The system is distinctly 'semi-ready,' and each engineer must use his own judgment in the choice of main groups in which he is particularly interested and in the further division of these for his own convenience.

Illustration I.—System suggested for a metallurgist particularly interested in copper and lead.

It is supposed that this individual is a subscriber to several technical magazines and a member of one or more mining or metallurgical societies. As one who is liable to move from place to place he does not wish to use a freight car for the transportation of his technical literature, so will plan to keep permanently only the bound volumes of the institutes, a few up-to-date text-books and those articles in the technical magazines which are likely to be of real use to him. He wishes to dispense with such articles as may become out of date, for without simplicity and real usefulness his system will soon become a burden. That is the one thing to be most feared, and he therefore remembers to avoid the cumbersome and complex, particularly in laying his plans.

He first has a carpenter construct for him, at a cost of a few dollars, a filing cabinet, consisting simply of a set of shelves 5 ft. $1\frac{1}{2}$ in. wide by 5 ft. $8\frac{1}{2}$ in. high, partitioned into 30 uniform pigeon-holes $9\frac{1}{2}$ in. wide, 12 in. deep, and $12\frac{1}{2}$ in. high, a size that will conveniently take a page from the oridinary technical magazine, either flat or on end. The outside measurements given allow for the construction to be of one-inch material, except the vertical partitions, which are one-half inch. The compartments are in five horizontal rows of six each. Above the first four rows

of these compartments are pasted typewritten slips bearing the following headings:—

COPPER

A. General Descriptive Articles.
B. Leaching.
C. Roasting.
D. Blast Furnace Smelting.
E. Reverberatory Furnace Smelting.
F. Converting.
G. Refining.
H. Miscellaneous.

LEAD

I. General Descriptive Articles.
J. Roasting.
K. Blast Furnace Smelting.
L. Other Smelting.
M. Refining.
N. Miscellaneous.

O. Other Metals.
P. Ore Dressing.
Q. Sampling and Assaying.
R. Power.
S. Transportation.
T. Fuel.
U. Refractories.
V. Financial Reports.
W. Management and Labour.
X. Miscellaneous.

The first two compartments of the last row are left empty, but will probably be used later for subjects which were not at first indexed separately. The last four compartments are used to hold society bulletins, and technical magazines received but not filed. When a technical magazine is received it is read and the articles worth keeping marked in pencil, X, XX, or XXX, according to the grading. It is then placed in the bottom row compartment reserved for such, and to it are added others that come in during two months. At the end of the second month the first month's magazines are removed—those for at least one month always being retained there—taken apart, the advertisements and articles not desired being discarded while those to be filed are all placed together. They are then ready to be numbered and indexed. If the articles are to be placed vertically in the cabinet the number should go on the upper left-hand corner; if flat, on the lower right-hand corner of each. First, the letter showing the proper compartment is marked on the sheet, followed by the numerical position of the article in its compartment, thus R23 would be the 23rd article filed under 'Power.' When all have been numbered the indexing is done.

The index consists of a small cabinet of drawers holding, say, 5,000 cards three by five inches. The first article is, we assume, a very good one on the North American Copper Co.'s Smelter, by John Jones, and describes that company's roasters, reverberatory furnaces, and converters. It has been given the number A17, and the notation XXX. It is then indexed under North, Copper Roasting, Copper Reverberatory Furnaces, Copper Converters, and, if desired, Jones. On the first card, under N, would be written: A17—No. Amer. Cu. Co. Smelter—XXX.

The next article is a short one describing the construction of tanks in electrolytic lead refining, marked M12, XX. This is not indexed under lead refining, as all the articles in compartment M are on that subject. An index card under T is inscribed: M12—Tanks in Pb Refin., Constr.—XX. An article on only one subject may not need to be indexed at all, the examination of its compartment locating it quickly when wanted. When two articles ordinarily filed in different places appear on the same page the sheet should be placed in the compartment of the more important article, and the index be made to refer to the other. Any society bulletins received during the previous month are examined, and the interesting articles indexed, the volume and page being marked on the card, but the articles *not* being filed. The bulletins are simply retained till the bound volume containing the articles arrives, when the former may be thrown away. One or two hours per month will probably be sufficient to file and index the valuable articles received by the average engineer and if the work is done regularly it will not become a burden. After such a system has been followed for five years the engineer should, once each year, go over his files, removing and looking over the articles for one year at the bottom of each compartment. Many articles will probably be out of date by then and may be thrown away. This practice prevents the accumulation of a great amount of dead material.

Illustration II.—System suggested for a copper plant with concentrator, roasters, reverberatory and blast furnaces, and converters.

This hypothetical company believes in providing for the use of its staff practically all the technical literature published in English on wet concentration and smelting. One good-sized

room is set apart as a library, and all the desirable text-books are kept on the shelves, accessible at all times to the men, but not to be removed from the building.

In the arrangement of the technical magazines two main courses are open. They will be considered separately.

(*a*) An elaboration of the system already described for the individual, articles being clipped, numbered and filed, with a comprehensive cross-index. As the number of articles to be filed is much greater—being taken from a greater number of magazines and including more minor articles—it will be best to choose a larger number of main groups, and to subdivide these, in order that the number of articles in any one division may be kept within reasonable limits. In this case numbers, instead of letters, must be given the main groups, but letters will be most suitable for the subdivisions, separating the numbers of the groups from those showing the position of the articles in the subdivision, thus 27R16, as mentioned above. In this case about 60 compartments should be provided for filing, these being 15 in. wide, 12 in. deep and 14 in. high to allow of the use of an envelope, placed vertically, to contain the articles in any subdivision. The following is a grouping which might be adopted—to save space the headings of subdivisions are not given except under crushing, this being simply to illustrate the method:

ORE DRESSING

1. General Descriptive Articles (arranged alphabetically under names of companies).
2. Crushing.
   - 2A. Theory.
   - 2B. Jaw Crushers.
   - 2C. Gyratory Crushers.
   - 2D. Disc Crushers.
   - 2E. Rolls.
   - 2F. Stamps.
   - 2G. Chilian and Huntington Mills.
   - 2H. Ball and Tube Mills.
   - 2I. Other Crushers.
   - 2J. Miscellaneous.
3. Screening.
4. Classification.
5. Jigs.
6. Tables.
7. Slime Treatment—Gravity
8. " " Flotation.
9. Mill Buildings.
10. Costs.
11. Miscellaneous.

METALLURGY OF COPPER

12. General Descriptive Articles.
13. Leaching—Solution of Metals.
14. " Precipitation.
15. Roasting in Multiple Hearth Furnaces.
16. Blast Roasting.
17. Smelting Blast Furnace Construction.
18. " " " Operation.
19. " " " Slags.
20. " " " Miscellaneous.
21. " Reverberatory Furnace Construction.
22. " " " Firing.
23. " " " Operation.
24. " " " Waste Heat Boilers.
25. " " " Slags.
26. " " " Miscellaneous.
27. Converter Construction.
28. " Lining.
29. " Operation.
30. " Slags.
31. " Miscellaneous.
32. Flues.
33. Disposal of Dust.
34. Blowers.
35. Bins.
36. Sampling.
37. Assaying.
38. Costs.
39. Miscellaneous.

GENERAL

40. Metallurgy of Lead.
41. " " Zinc.
42. " " Gold and Silver.
43. " " Iron and Steel.
44. " " Other Metals.
45. Power, Electric.
46. " Steam.
47. " Gas Engines.
48. Transportation by Rail.
49. Belt Conveyors.
50. Fuel.
51. Refractories.
52. Financial Reports.
53. Management.
54. Labour.
55. Safety First.
56. Miscellaneous.

It will be well to apportion the technical magazines to different members of the staff, and make them responsible for marking the articles to be indexed. The actual work of indexing and filing should be done by a junior, who may have to spend up to one day per month on this task.

The work of indexing should in this case be carried out in greater detail than for the individual, though on the same general plan. As it will seldom be necessary to move the filing cabinet, every article which is at all likely to be useful should be saved. The sheets may in filing be placed in large envelopes, each containing all the articles in one subdivision, and opening outwards for easy access, or the subdivisions may be separated by projecting cards. The system of numbering is the same as that in Illustration 1, the highest numbers in each section showing the latest publications.

(*b*) The advertisements are removed from magazines and the reading matter is bound; binders being usually supplied by the publishing houses at small cost. In this way the filing cabinet is required only for occasional pamphlets, and the card index becomes all-important, although the indices of the publishers cover the bound volumes separately. As in the previous illustrations, the index includes articles in the transactions of the technical societies. This system is the one best adapted for use in the mining schools. Mr. Keeney, in the paper referred to above, expresses the opinion that when much literature is being indexed the cards should be arranged according to the subjects of the Dewey system, but the alphabetical system will probably appeal with greater force to the average engineer.

Illustration III.—System suggested for the filing and indexing of mechanical drawings.

Conditions are in this instance almost ideal for filing, and, as the drawings will usually be referred to over a long period of time, it will pay to do the work more thoroughly than in the previous illustrations.

The drawings are best kept in shallow drawers, of which a sufficient number is available to allow of a maximum of 100 sheets per drawer. A classification of the drawing is first made according to subject, the latter being chosen so as to keep the number of drawings under each subject within the capacity of a drawer. Then the drawings in each group are rearranged according to minor subject. The major groups mentioned are numbered consecutively, the minor groups are lettered, and the drawings in each minor group numbered, chronologically. This

arrangement is the same as that given under Illustration II (*a*), so that 15R7 refers to drawer 15, sub-group R, and drawing 7. As new drawings are made they are thus easily placed in their proper position. The usual card index is required, and it is also convenient to make a list of all drawings in the order in which they are found in the drawers, with sufficient spaces left for the insertion of new headings. The time required to file the drawings thus and keep up to date a list and index is trivial compared with that expended in looking for drawings kept without a good system. With each drawing lettered and numbered according to its exact position very few will become misplaced.

A PLEA FOR DEFINITE TRAINING FOR SOCIAL RESPONSIBILITY, BY MEANS OF OUR EDUCATIONAL INSTITUTIONS.[1]

By C. V. Corless.

Annual Meeting, Montreal, March, 1917.

In previous communications[1] to the Institute, the writer discussed at considerable length some of the causes of the general antagonism between capital and labour, dealt with this condition as one problem (certainly one of the most fundamental) in our social development, and gave strong reasons for his belief that the most successful mode of attack of these social problems will prove to be through our educational institutions. A brief outline of suggested educational reform, looking to this end, was given and a suggestion, as to the quickest and most effective method by which this reform might be instituted, was outlined. Since these articles were published, the writer has received a number of letters indicating very deep interest in the discussion and expressing these correspondents' belief in the importance and practicability of the suggestions therein made. The whole question is of so great importance that no apology is needed for continuing the discussion.

The general shaking up caused by the war, both to our ideas and to society itself, more especially in Europe, has caused many revisions of accepted opinions regarding social as well as many other questions. When a man, for any occasion, has need to rouse himself to his full strength, for some maximum exertion, he very quickly discovers any unknown defects in his constitution and is suddenly reminded of any known weaknesses, which may have been forgotten in ordinary days of comfortable existence. When the exertion is past he will do well to give such attention to these defects and weaknesses as will restore him to perfect health and soundness of constitution. This analogy applies very closely to the social organism. Are we going to drop back into the old groove, when the war is over? Or, are we really determined to face social issues squarely,

[1]C.M.I. Bull. No. 53, Sept., 1916, p 764; *see also* C.M.I. Bull. No. 56, Dec., 1916, p. 1035, and Bull. No. 57, Jan. 1917, p. 43.

without prejudice from apparent personal interests; to consider these problems with the same degree of concentration as we apply to our professional difficulties; and to use our full influence, not only to secure palliatives, but to bring about that wide-spread interest in, and study of them, that is our only hope for discovering and effecting their progressive solution?

Society has permitted the development of a system of industry that creates millionaires and paupers, with all grades between; that segregates thousands of people in spaces that hundreds, or better still, tens, should occupy; that is frequently more careful of its material capital than of its human capital; that attaches greater importance to its material, than to its human product; that more and more concentrates the ownership of industrial capital and the control of most of the necessaries and luxuries of life into the hands of a few; that has, under one disguise or another, substituted in many cases a degree of servitude, without responsibility of the masters, for earlier forms of serfdom or slavery with their responsibility for the care of those who served. We have developed a social condition in which, to a large extent, privilege and responsibility have become dissociated; in which rights rather than duties are mainly insisted upon; in which conduct is to a far too great extent restrained by law and police, rather than by ethical principles; in which stealing a morsel of bread, in order to maintain mere existence, may lead to imprisonment, whereas stealing millions, by methods of high finance or of predatory business, is often condoned and frequently leads to social preferment; in which, more and more, the gilded city and the movie are, by the many, preferred to the realities of rural life and the beauties and attractions of nature when viewed with intelligence. We have evolved an educational system in which an attempt is made to develop the growing minds of the young by feeding them almost exclusively on the husks and symbols of knowledge—language and number, whereas the absolute necessity of training the powers of observation and of forming the habit of independent investigation, discovery and reasoning, as a basis of real mental growth and vigour, is almost entirely overlooked; in which, for higher training, knowledge of a dead language is made compulsory, whereas an understanding of our living

social conditions and of the causes underlying them is ignored; in which we are only now beginning to realize that it is one of the highest duties of the State to train its citizens to become efficient producers of the world's needs, whether of goods or of services, whereas we have not even yet as a people come to realize that the State owes to its citizens the still higher duty of giving them an early training in the duties of efficient citizenship. What steps can be taken in the direction of bettering these social conitions? Any real improvement will at best be gradual. How can we lay a sure foundation for effectively accelerating such gradual improvement?

Scientific and industrial research, state-aided, will certainly add to the nation's wealth and is therefore highly important. Wide-spread and intensified vocational training will add enormously to industrial efficiency and therefore to national wealth, and hence is to be encouraged to the utmost. Conservation of our natural resources, the lessening of national waste, is again conducive to the same end and is therefore fundamental to our prosperity. But will all of these great movements, though of vast importance, strike at the root of the matter? Consider a moment. Was it not industrial advance, based on scientific research, that has resulted in present social conditions—the extremes of wealth and poverty, the segregation of large populations into small spaces, the continual war of labour and capital, the false ideals, with the attendant evils of all of these? Has not the engineering training of our vocational colleges merely accelerated the industrial advance, without assisting to correct the resulting social conditions? Are not scientific and industrial research, vocational training, conservation and all other similar movements, of vast importance as they are, all in a direction merely to increase material wealth without improving its distribution? Unless *pari passu* with these advances, we improve in other respects, shall we not merely further widen the chasm that already yawns between the two extremes of wealth and poverty and thus intensify the evils of the existing and increasing social stratification, with the labour antagonisms and other evil by-products engendered by it? All these advances are but parts of a mighty economic engine constructed by society for the production of material wealth. But they do not provide the

engine with an effective governor. An engine so built and so run will merely race to its own destruction. If, parallel with vocational training, we introduce into our educational programme an effective compulsory course in social (including economic) training, we shall at the same time perfect a governor for the engine and, in place of racing, our engine, under proper control, will become a most efficient instrument of service to society as a whole. Some preparation will thus be made for meeting the menace of uncontrolled distribution of the wealth produced by human energy.

A statement in the *Mining and Scientific Press* of January 20th, 1917, by Saunders, may throw light on the extent to which such conditions are being created in the United States. Similar conditions prevail in Canada. He says: "During the past 25 years the population in the United States has increased by about 70%, while the important basic industries have been enlarged at a rate of increase between 500 and 600%". This, as is well known, has led to a great increase in national wealth and in the number of millionaires. Has it appreciably lessened the percentage, or improved the condition, of the slum-dwellers? As stated above, we admit the importance of scientific mobilization of our material resources for material advancement, also for defense. There is no contention against this; rather the reverse. But we do maintain that great emphasis on the ways and means of securing material progress only, unless accompanied by equally great emphasis on those planned to secure social progress, will not only fail of the highest material results but will prove to develop, along with such material progress, the forces that will effect its own disintegration and final destruction. Strikes and lock-outs are manifestations of these forces. We do maintain that scientific and industrial research, vocational training and conservation of natural resources, taken alone, are insufficient to attain true national progress. Vocational training has its eye on material advance. Social training has its eye on human welfare. Both are necessary for the highest results from either. Each is necessary for the best results from the other. If we make our educational system responsible for the one, why should we not make it equally responsible for the other? Why should we allow these com-

plementary aspects of a well-balanced educational programme to become divorced?

Every reader of history is aware that, notwithstanding the horrors of the present war, altruistic feeling has, on the whole, steadily increased in Western civilization. The disappearance of gladiatorial fighting and of animal baiting for sport; the abolition of slavery as an institution; the more and more humane treatment of criminals; the steady increase of such social legislation as safety, compensation, insurance and pension laws; the spread of free schools; the organization of such philanthropic institutions as hospitals, orphan homes, and asylums for physical and mental defectives—all these point in one direction. All such progress indicates a growing sympathy for the young, the stricken, the less favoured and the helpless. These manifestations of growing altruism and the evidence of a spreading belief in some form of socialism indicate a growing hunger for knowledge and understanding of social problems, to satisfy which, no present systematic attempt is being made by means of our educational institutions. We believe that the most efficient means of increasing the rate of real social advance, and of counter-acting the spread of utopian theories of social progress by directing into more practical channels the energy now spent on their propagation, will prove to be, as herein advocated, a thorough general foundation in the social sciences, laid by our educational institutions.

This method of definitely preparing for sound social advance has the great advantages of being immediately practicable, of avoiding prejudiced alarm and of securing progress by development and forestalling revolution. It prepares individuals definitely for entering into social relationships and thus lessens the restraints of laws and police. When once the educational programme of social science has been introduced all down the line of our educational system, society will become thoroughly permeated with rational ideas of social progress, public opinion will become gradually altered, and further necessary social legislation will be only a natural expression of the desires of the people. This is the true method of progress in a democratic country.

It is not maintained that any single group of people, or

any one class, or even any restricted number of classes, may be responsible for prevailing social conditions. We are all the product of general social conditions, inheritors of the past, with views whose foundation was laid by the prevailing educational system of our time. We little realize how greatly the centres of interest formed during the period of our education, influence the lines along which our adult reading, study and thought will be continued. This is frequently expressed in the aphorism, "As the twig is bent so is the tree inclined." On reaching maturity we enter into, and are carried along by, the current of present day activity and struggle, with but little opportunity and no educated tendency, to try to obtain a clear and deep understanding of the social conditions surrounding us, in the actions and re-actions of which we are taking part. We merely accept them as being a necessary condition of our existence and enter into the general struggle with no clearly defined principles or responsibilities. We look to personal, not to social, improvement. No clearly defined effort was made in our period of adolescence when we were forming mental centres that would determine to a large extent our future interest in and attitude toward most questions, to prepare our minds for present and future social responsibility. A golden opportunity was missed. How can this condition be improved most quickly? To the writer it seems almost axiomatic that the work can most quickly and most efficiently be undertaken by our educational institutions.

Consider for a moment the effect of early education in a given direction. If we go into a large library, which alcove do we make for? If not seeking mere amusement, nine out of ten of the members of this Institute would be found in the technical section and probably eight of the nine in the mining or metallurgical corner. Even when we travel for pleasure many of us involuntarily make for a mining district. Or, if not, we head toward some other centre of interest formed early in life. These illustrations will serve to make clear the tendency to grow in directions fixed early in life, to accumulate information around existing centres of interest. Seldom are absolutely new centres of deep and lasting interest formed after we reach full maturity, after we enter the rush and turmoil of busi-

ness life. The bending must be done during adolescence, while we are still twigs. We may later grow to any extent in the direction toward which we are then inclined. Does not this simple illustration suffice to indicate the vast importance of making certain that the great question of social progress, heretofore almost entirely neglected as a clearly defined part of our general educational programme, should form one of the centres of interest of prime importance in the growing minds of the rising generation?

We have been trained under an educational system practically ignoring social science. Our masters were practically all similarly trained and had no clear and adequate knowledge of social and economic questions. Is it any wonder that we have reached our present period of life with views not clearly defined on these important and highly complex questions? The war has forced our minds out of old grooves. We now realize that here is a vast field of the very first importance, regarding which, even most of our legislators, who received like training to ourselves, have most inadequate knowledge. They can only either blindly experiment or, as more generally happens, avoid the issues. While urging more widespread vocational training with great vigour, we are losing sight of the fact that mere increase of material wealth, with the practical exclusion of social considerations, will only add to existing evils. We even more urgently need educational reform in this direction. With the two reforms, the one looking to individual, and the other, to social efficiency, we shall not only be developing a productive engine of great power but we shall also have taken care that the governor is properly designed and regulated. A democracy individually efficient and socially trained and efficient must be one not only of wealth but of happiness and contentment.

For the sake of clearness and definiteness, let us assume that the programme outlined in a former communication[1] could be brought into operation at once. What results should follow?

In the first place, every graduate of our Universities within five years' time would enter into private business or public life, with a trained social consciousness and a clear grasp of

[1]C.M.I. Bull. No. 56, Dec. 1916.

ethical principles as a guide to his conscience in the consideration of social relationships. Every graduate would have such training and such centres of interest as would enable him to weigh intelligently the probable social, economic, civic and political consequences of such a change as, let us say, the granting of a reasonable living wage to the many thousands of workers on this continent who cannot under present conditions, by any possibility, bring up their families in decency and self-respect. He would also from his ethical training have a rational basis of principles for the guidance of his conscience as to where his responsibility lies in relation to the class of problems of which this is typical.

Within say the next five years our schools of pedagogy, normal and high schools would begin to turn out students with similar, though not quite so thorough, training; and within say five more years—say 15 years in all—even our public schools would begin to be filled with teachers having a clear knowledge of the fundamentals of the sciences grouped under the term Sociology, such as would give them an entirely different view of the importance, the possibilities, and the responsibilities of their work and an altogether new view of their own social responsibility.

Within say 20 years at most the whole scheme would be in smooth working order from top to bottom of our educational system. At the same time, a similar change in mental attitude toward social problems would have begun to permeate society through all the other professions receiving training at our educational fountains—law, medicine, theology, engineering, finance, accountancy, business management, politics. Practically all our leaders of thought and action would have received such social enlightenment during their period of preparation as would substitute thorough knowledge of, and deep interest in, all important social questions, for the present inadequate knowledge and prevailing indifference regarding these questions.

At the same time, our lower schools would more and more find it possible to impart to those students of 13 to 16 years who must finally leave school then, such essentials of social knowledge and responsibility as constitute the

basic facts on which society rests. It would of course be absurd to expect to impart at 14 or 15 years a thorough knowledge of economics, of civics, or of ethics. None the less, the teacher with clear understanding of these, deeply interested in social progress and thoroughly impressed with his responsibility to society, will find that such a foundation in social knowledge and ethics can be laid, even with these immature students, as will lead them to continue their interest in these questions in later life, to recognize their place in the civic body, and thus to become more intelligent and efficient citizens than under the present system of training.

In the meantime, our universities, in close co-operation with our governments, would be undertaking intensive investigations of the entire range of social phenomena and would be finding, as years pass, a wider and wider circle of readers, and a deeper and deeper interest displayed by the public, now generally trained to take an intelligent interest in these questions.

Will you venture to say that such broadened, deepened and quickened public interest would not result in furthering social justice and improvement? Can you mention any other institution that so gets at the root of the matter as our educational system, properly directed?

This educational preparation for solving social problems and for turning out more intelligent and efficient citizens does not presuppose any particular social theory such as, for example, any modification of socialism. It rests on the belief that existing social ills have mainly arisen from general social indifference; this in turn has been a result, partly of over-emphasis of the importance of material wealth coupled with failure to realize the importance of the human factor, partly of the complexity of the problems involved, partly of human inertia and general selfishness and partly of a number of historical causes. It rests, further, on the belief that this class of problems, in which the interests of all or nearly all are involved, can be solved only by democratic means, that is, by the enlightenment with regard to them, of large numbers of people—the majority, in fact. It also assumes that, with so many conflicting interests involved, an effective solution can only be found by such a general training

as will lead to a widespread concentration of mental effort on the many diverse factors involved. This necessitates the reversal of our educational policy hitherto followed regarding social science, which has been simply a policy of omission. Social problems can never be solved by neglect; nor can they be successfully solved in spots, as at Ford, or elsewhere; nor can they be solved by alleviating symptoms, through charity or otherwise; nor will they be solved finally without raising the general average social and ethical consciousness of the people and altering the fundamental conditions that give rise to the problems.

Increased knowledge of social conditions, understanding of necessary social relationships, ethical sensitiveness, are all fundamentally necessary to the solution. The institution most readily at hand and most easily able, or that can most easily be endowed with ability to undertake the work, is our educational system, every department of which, from university to common school, is needed. Time will be required even if we act quickly. How can we so reach the public as to lead to the necessary demand for this important educational reform? If undertaken at once, we should see a great change for the better, within the span of life of most of us now living. I well remember the introduction of temperance teaching into our schools and into those of many of the States. Today, even omitting the influence of the war, we have nearly attained to prohibition on this continent north of Mexico. The effort at the time was very feeble, very forlorn—but the seed germinated, took root and is maturing into an abundant harvest, nevertheless. How vast will be the harvest if we can effect this seed-sowing? None of us can begin to estimate the importance of the results that would be attained, even within a single generation. Seed sown by means of our educational institutions is cumulative in its results. The harvest of social progress resulting from it will be both continuous and increasing. The foundation would be laid for a progressive solution of existing social problems and for a continual study and understanding of new social adjustments as the necessities arise.

We are members of a democracy. This is merely another way of saying that, if our social organization and institutions

are to be improved, it can only be done by society itself. No superior or beneficent government or other power will accomplish this for us. Our government and its laws are the expression and reflection (sometimes inadequate enough) of our own knowledge and wishes. These are platitudes of which we need reminding. Hence society can only be improved from within. A wide-spread and adequate understanding of social problems is the prime requisite, (1) for finding their solution and (2) for effecting their solution. He who now, wittingly or unwittingly, takes advantage of the general economic ignorance that prevails and secures an unjust share of the world's goods and who then makes amends by big philanthropy, would not find the opportunity and would probably not have the desire, in a society trained to broad and deep social consciousness and sensitive to the principles of social ethics. This general condition of mind must be made a central aim of our educational system. The most efficient citizen is not he who is most efficient industrially only, but he who is also fully aware of the just rights of others and recognizes them; who fully realizes his interrelations with other members of society; who knows that his personal welfare is inseparable from that of his fellows. As well may the hand smite the foot or the eye, as a man be unjust to his fellow man. We need a complete change of mental attitude toward social questions. Our much-lauded 'individualism'—an expression of merit in one aspect, but a mere euphemism for utter selfishness in another—has made us socially indifferent. It is a result of our education. The educational system that produced it must be so modified as to replace social indifference by effective social knowledge. Vocational training alone will only exaggerate the 'individualism.' Vocational training makes for efficient individuals. It must be supplemented by the social training we are advocating as a corrective. This will give us not only efficient individuals but efficient citizens. To increase individual efficiency by added general vocational training will merely exaggerate existing social evils. Now is the vital time. While introducing the industrial, the vocational half of a practical educational programme, as we are doing, let us not fail to introduce the humanitarian half that will lead to intelligent and efficient citizenship.

The aims of education should be not only to develop the young as individuals but to prepare them as well to relate themselves intelligently to their physical, their industrial and their social environment. If we lose sight of the last of these aims, as we have been doing to a great extent, how can we hope for any other result than social inefficiency and confusion. The natural result must be men with a very low feeling of responsibility toward society and toward the State. Here we place our finger on the greatest single defect in the present educational systems of our country.

Even to mere industrial advancement, there are two fundamental sets of conditions—material conditions and human conditions. On the material side, prosperity is based on growth of inventions, increase of real capital, (that is, the apparatus of production), accumulation of savings and abundance of raw materials. On the human side, it is based on the physical health, intelligence, morality and social welfare of the workers. Looking at the question nationally, it must be self-evident that even material prosperity is lessened, if any considerable percentage of the population is kept at or under the bread line; if they receive so small a share of the results of productive effort as to lower health, intelligence or self-respect. But this is today the well known condition of large numbers on this continent, many of them in the absolutely hopeless state of mind of slum-dwellers. Our social-consciousness has not yet been developed to the point at which this problem is being seriously tackled on any broad basis. The quickest way to increase social consciousness is definitely to introduce a thorough social training into our educational system; to carry out the most extensive researches through the co-operation of our Governments, Dominion and Provincial, and our Universities—under the direction of the latter—expanded and made adequate for the work. All such researches should be kept out of politics, so that they may be carried on as searches after truth, which is impossible if there is political distraction. The propaganda, which will widely disseminate the results of these researches, through our educational system, will secure the necessary political support for such legislative steps as research proves to be advisable. In a democratic country, this appears to be the surest and shortest way to

secure definite and certain advance both in social consciousness and in necessary social legislation. We thus secure not only the necessary laws but the disposition to obey them. Our educational system, properly conceived, is the greatest single civilizing force we have. To secure most rapid social advance, we should no longer practically ignore the possibilities of our educational system in effecting it.

Ultimately society must decide what is fundamentally a just distribution of wealth; how much interest and rent capital is entitled to; what is a just return to the business man, as profit, for the risk undertaken and enterprise shown; what is a just return to labour; what are the just rights and relationships of the public in relation to the individual business enterprise; how far these questions can be answered practically, without removing the incentives that are the psychological foundation of material progress. Conceal the issue from ourselves, or postpone the issue by continued social indifference as we may, we do well to recognize the ultimate goal and to measure the value of any tentative forward steps by their progress in this direction. Any real progress toward industrial peace and social justice will be toward this goal, unless indeed, in the meantime, society discovers some better and more practical system than that of private property with universal competition, which seems at present unlikely.

When all interests are considered, is it possible that real progress toward settling the just distribution of wealth should be made without some form of regulation by public authority? In a democracy, can any better preparation for such progress be made than by a thorough general training of all our leaders (educational, professional, industrial and social), in economics, public responsibility and social ethics? A shallow training will merely increase the numbers of demagogues and agitators, with all the cocksureness of those who have never even discovered the complexity of the problems for which they offer a ready-made solution. At the same time, every effort must be made by teachers and other leaders, thoroughly trained therefor, to impart to every member of the community a deep feeling of social responsibility and to increase not only true social consciousness but ethical sensitiveness. All must somehow be

brought to feel that the fundamental basis of present society is the interdependence and responsibility of its parts; that the parts of the social organism are as interdependent as the organs, nay, even as the very cells, of the human body; that rights and responsibilities are inseparable. The institution in our country that most definitely influences the ordinary man, both directly and indirectly, is our educational system. Here is a great humanitarian opportunity that should no longer be ignored.

Why, it may be asked, have we left the discussion of the problem of labour and capital and dealt with social problems in general? For those who have not followed this discussion in its earlier stages, I would merely say briefly that the age-long industrial dispute is merely one aspect of the broader question of the just distribution of wealth, which problem is fundamental to most social questions, all of which arise from the failure of our social evolution to keep pace with our material advancement. The antagonism between capital and labour is not an isolated question. It arises from a more fundamental general condition. It is symptomatic in nature. If you went to your physician suffering from a headache, he would probably look you over thoroughly and might finally decide you were suffering from eye-strain, from indigestion, or from some specific disease. He would not, in any case, attempt to cure the headache as such. True, he might give you a headache powder in order to relieve the symptom. Our previous examination of this problem[1], had not proceeded far before it became clear that this dispute is merely a symptom; that it arises fundamentally from lack of social knowledge, from want of that deep understanding of social questions that must be based on thorough knowledge and above all from deficiency of ethical feeling. If social problems were widely and deeply understood and if the fundamental training of all led to ethical sensitiveness, then the cure of this, as of other social ills, would quickly be effected by democratic means. Our central aim should, therefore, be to use every available educational means, omitting none however humble, in order to ensure advance toward this condition of human society.

[1]Bull. C.M I. No. 53, Sept., 1917.

This does not mean that, while working in this direction, it is undesirable to take any possible palliative measures. By all means, let us do so. Such measures the writer has left to others to deal with, as not being within the scope of his discussion. But for the sake of clearness and for real progress, it is necessary to distinguish clearly when we are attempting to alleviate the symptom and when we are working in the direction of removal of the cause of the social ill.

Hitherto, our social adaptations to meet altering material conditions, have been largely haphazard, blind and unconscious. Our plea is that our social progress be made orderly, purposeful and conscious, which method only is worthy of men who have reached our advanced stage of civilization in other respects. To accomplish this, we believe we have proven it to be necessary that the definite study of social questions should be made a fundamental part of the education of every citizen during adolescence. How else can this end be definitely reached? If this is correct (a view that appears to the writer to be beyond controversy) a beginning can be made at once. We have in our educational systems of the various provinces, machinery ready and in operation. We have, within easy grasp, the possibility of effecting a silent social revolution of inestimable importance. The greatest present problem is how to gain the ear of a sufficient number of the public in order to bring adequate pressure to bear on the proper authorities to effect the necessary educational reform.

The fact that a technical institute such as this should set aside an entire session for a discussion of questions of this nature, is of deep significance and is reason for great encouragement. This discussion indicates that the members of this Institute are becoming more alive to the fact that the consideration of technical matters only, does not cover the full scope of the mining business or of any other business. It indicates also a deepened humanitarian feeling brought about mainly by the suffering, sympathy and loyalty, caused by the war. Our action is typical. Similar discussions are taking place in the press generally and in most other organizations. All this indicates a very widespread readiness to take definite steps toward the progressive solution of social problems. The most

serious question for us to consider is: How can we help secure that unity of public discussion and action which alone can bring concrete results?

DISCUSSION

Mr. JOHN A. DRESSER: This paper is one that should not be passed over without at least a word of commendation, even if to discuss it at this time is beyond us. Mr. Corless has laid out for us work for the next twenty-five years or more. That remark is not meant to be derogatory, but quite the contrary.

The PRESIDENT: On behalf of the Institute I wish to express to Mr. Corless our very great appreciation of his thoughtful exposition of this most important subject. As Mr. Dresser has remarked while few of us feel qualified to discuss these great sociological problems, we are all nevertheless giving thought to their solution, and this general recognition that the problem exists and must be faced and solved is bound before long to lead to definite action and important results.

Professor J. W. BELL: I consider this paper to be one of the most important ever presented to the Institute, and I have much pleasure in moving the following resolution:

"RESOLVED that the Canadian Mining Institute endorses the educational reforms outlined in this paper, and recommends them to the careful consideration of the Educational Departments of all the Provinces of the Dominion; and that copies of the paper and of this motion be sent to the Ministers of Education of each Province in the Dominion."

The resolution was seconded by Dr. J. B. Porter and adopted unanimously.

GENERAL INDEX

NOTE—The names of authors of papers are here printed in small capita letters and the titles of papers in italics.

A

PAGE

Adams, L. D. 167
Africa, South—
  Coal trade. 434, 438
Ainsworth Div., B.C.—
  Ores, treatment of. 109-117
Alaska—
  Kelp. 308
Albert Lake, Oregon. 312
Alberta—
  Bituminous sands. 447-459
  Coal, analysis. 425
  " low grade, utilization. . . . 445
  " pulverized, locomotive tests. 427-430
  " reserves. 432
  " resources. 436
  Refractory clays. 229
Alex. Fleck, Limited, Ottawa. . . . 278
  Sand tests at foundry of 278-286, 287
Algin. 310
Alice Arm, B.C.—
  Molybdenite. 129–131
Algoma Steel Corporation—
  Mines and plant of. 339, 344
  Notes by Irwin. 375
  Products. 346, 347, 349
  Siderite treated by. 379
Algonquin Lake—
  Moulding sand. 267
Allen, Carl A. 481
Alloys, *see also* Ferro-alloys—
  Aluminium. 474
  Magnesium for. . . 472, 474, 476-480
A Plea for Definite Training for Social Responsibility, by means of Our Educational Institutions: CORLESS. 491-506
Alsace—
  Potash. 294, 301
Alumina—
  Combination with zinc oxide. . . 172
  From alunite. 307
Aluminium—
  Alloyed with magnesium. 476
  Power for manufacture of. 461

PAGE

Aluminium dust—
  Price of. 6
  Silver precipitation with. 6
  " cost. 10
Alunite—
  Aluminum from. 307
  Potash from. 306
Amalgamation—
  Ainsworth div., B.C. (silver) 113, 115
  McIntyre mine, Porcupine dist. 94
America, South—
  Potash. 301
American Association for Testing Materials. 159
American Cyanamid Co. 466
American Trona Co. 311
Ammonium molybdate. 133, 134
Anaconda Copper Mining Co. 135, 136
  Electrolytic zinc. 135-136
  Zinc refining practice. 157-160
Analyses—
  Coals for pulverization, Canada 864, 865
  Iron, copper ore, Tassoo Harbour, B.C. 397
  " foundry iron. 395
  " roasted, Magpie mine. . . . 332
  Magnesite, Atlin, B.C. 246
  " Austria. 237
  " Canadian. 239, 243
  " Greece. 238
  Magnetite, British Columbia. . . 396
  Moulding sand. 272
  " " Brockville. 286
  Silver-gold, Tepic, Mexico. 263
  " precipitate, Nipissing Mine. 9
  " Silver Glance Mine, B.C. 110
  " Silver Hoard mine, B.C. 112
  Zinc, Sullivan mine. 162
Antimony—
  Alloyed with magnesium. 480
Argenteuil Co., Que.—
  Magnesite. 240
Argentite—
  Amenability to flotation. 36, 37
Armour Fertilizer Co. 306
Armstrong, C. J. B. 18
Aschersleben Kaliwerke. 296

PAGE

Ashcroft, B.C.—
Mining near 118
Asphalt, *see also* Bituminous sands—
United States production 448
Assays—
Flotation concentrate, Buffalo mine 25
Lead, Kingdon mine 184
Zinc ore, Sullivan mine 172,173
Assiniboine Valley, Man.—
Clay 229
Athabasca river, Alta.—
Clays 229
Atikokan Iron Co.—
Furnaces of 344
Atlin dist., B.C.—
Magnesite 234, 235
" analysis 246
Atomizer (for lubrication) 409
Australia—
Coal trade 434–438
Magnesite 239
Austria—
Magnesite industry 237–239, 244, 245, 247
Austria-Hungary—
Coal trade 437–439
Potash 295, 300

B

Baker, Prof. M. L.—
Discussion by, on refractory materials in Canada 235
Barcelona, Spain—
Potash 301
Bates, Dr. J. S. 38, 78
Bauxite 307
" not found in Canada 235
Bear Lake, Ainsworth div., B.C. .. 110
Beauharnois, Que.—
Potsdam sandstone 233
Beaver mine, Cobalt, Ont.—
Mill 29
Belgium—
Coal trade 437, 438
Belknap, J.—
Discussion by, on Canadian iron and steel industry 380
BELL, PROF. J. W.—
Notes by, on rock crushing 108
Resolution *re* educational reform moved by 506

PAGE

Belleville, Ont.—
Ferro-molybdenum 126
Bell Island, Newfoundland—
Iron 398
Benson, Prof. H. K. 67
Bessemer iron ore—
Price of 362
Bethlehem Steel Works—
Potash recovery tests 304
Bibliography—
Zinc electrolysis 137, 138–158
Bituminous Sands of Northern Alberta: ELLS 447–459
Black Hills, S. Dakota—
Cyaniding practice 207
Blast-furnace *see also* Furnace—
Capacity of, in Canada 381
Potash from gases of 304, 305
Blasting—
Crownest dist., coal mines 406
Bleicherode, Germany 294
Boberg and Testrup 310
Bodaibo, Siberia 197
Climate 197, 205
Boiler compounds 410
Boilers—
Pulverized fuel for 430
Bone ash 231
BOYD, JAMES A.—
Paper on lubrication of mine machinery 409–415
Brazil—
Iron ores 398
Bruce Mines, Ont.—
Flotation process 78
Buckingham series—
Magnesite deposits 254, 255
Buffalo mine, Cobalt, Ont. 11, 28
Flotation concentrate 20, 33, 34
Mill 29
Oil distillation 53
" flotation tests 74, 78
Briquetting—
Coal, cost of 417
Iron ore 390, 392
By-product coke, *see* Coke—
Brown potash process 316, 317
British Columbia—
Coal, analysis 425
" reserves 432
Iron and steel industry, prospects in 389–399

PAGE

British Columbia—Con.
Kelp ... 308
" potash from ... 310
Magnesite ... 246
Molybdenite ores ... 131
Refractory clays ... 230
Broadbent, R. L ... 251
Brockville, Ont.—
Moulding sand deposits ... 276–278
analysis ... 286, 287
map ... 280
photomicros ... 282
Bresee, T. H ... 280
Bridge river dist., B.C.—
Magnesite ... 235

C

Cables—
Lubrication of ... 413
Cadmium—
Alloy with lead and magnesium. 478
Reduction of in electrolytic zinc. 160
California—
Bituminous sands, treatment ... 450, 456
Kelp ... 308
Magnesite ... 239, 247
Potash ... 310, 311
California Trona Co ... 311
Callow, J. M.—
Min. Ref ... 30
Paper by, and Thornhill on flotation practice in Cobalt dist.. 28–37
Callow flotation process ... 11
Cobalt dist ... 28
Buffalo mine ... 17, 28
Calumet, Que.—
Magnesite near ... 240
Canada—
Coal reserves ... 432
" trade ... 435, 436–438, 445, 446
Co-ordination for national service needed ... 3
Electro-chemical industries. .460–467
Iron and steel industries ... 333–388
Magnesite industry ... 237–255
Molybdenite ... 121–128
Power, cost of ... 468
exported to U. S. ... 469, 470
Refractories ... 226, 237
Resinous wood, oil from ... 65
Canadian Dept. of Mines.—
See Dept. of Mines.

PAGE

Canadian Furnace Co.—
Blast Furnace ... 340, 344
Canadian Mining Institute—
Activities of, 1916 ... 1
Metallurgical section ... 2
Canadian Potash & Algin Co. ... 310
Canadian Tube & Iron Co., Montreal ... 355
Canadian Wood Molybdenite Co.—
Mine ... 126
Production ... 128
Canadian Wood Oils for Ore Flotation: Parsons and Gilmore. ..38–93
Canadian Zinc Products Co ... 162
Capital and Labour—
Solution of problem ... 491–506
Carbonic acid gas—
Manufacture of ... 240, 242
Carnalite, *see* Potash—
Carpinteria, Cal.—
Bituminous sand ... 450
Caustic soda—
Use in flotation process. 42, 60, 71, 73
Cement—
Potash recovered ... 303
Chalcopyrite—
Flotation tests ... 52
Highland Valley, B.C ... 118, 119
Charcoal iron—
Mfg. of at Deseronto ... 380
Chat Island, Fitzroy tp., Ont. ... 181
Chili—
Potash ... 301
Chloride of magnesium—
Uses ... 244–246
See also Magnesite.
Chloridization—
Flotation concentrate, Cobalt dist ... 20, 34
Chrome brick ... 230
Chromite—
Canadian occurrences ... 230
City Street Improvement Co., Santa Cruz, Cal ... 454
Claudet, H. H.—
Paper by on concentration and marketing of molybdenite. 121–134
Clay (refractory)—
Nova Scotia ... 227
Claybank, Sask.—
Refractory clay ... 227, 229

PAGE

Clayburn, B.C.—
Refractory ware..... 227
Clinton Mining Division, B.C.—
Chromite..... 231
Coal—
Canadian fuel situation.....443–446
Crowsnest dist., Alta.....400–408
Empire's trade in.....434–446
For iron and steel industry..363, 364
" " British Columbia..... 389
France, consumption..... 467
Power from..... 466
" conservation..... 467
Pulverized, for locomotive fuel..
416–433
Coaling stations—
Pulverized fuel..... 420
Cobalt dist.—
Flotation oil experiments..38, 70–74
Flotation practice in.....28–37
Milling practice..... 11
" " flow-sheet..... 29
Cobalt Reduction Co., Cobalt, Ont.—
Mill..... 29
Coe and Clevenger..... 220
COFFIN, J. S.—
Paper by, on pulverized fuel for locomotives.....416–433
Coke—
By-product, Nicola dist..... 395
" notes by Davidson
393, 394
" notes by Haggen.. 394
" Vancouver Island, cost..... 390
Canadian fuel problem, use of.. 445
COLE, ARTHUR A.—
Acknowledgements to.....38, 78
Presidential address..... 1–5
COLE, L. H.—
Paper by, on occurrence and testing of moulding sands....265–291
Compressors—
Lubrication of..... 412
Concentrate—
Flotation (Cobalt dist.), treatment..... 33
Concentration—
Buffalo mine.....15, 16
Cobalt dist., compared with flotation.....35, 36
Galena, Ainsworth div., B.C... 116
Iron.....390, 391
Kingdon lead mine..... 183

PAGE

Concentration—(Con.)
McIntyre mine..... 94
Molybdenite.....121–134
Concentration and Marketing of Canadian Molybdenite: CLAUDET
121–134
Coniagas mine, Cobalt, Ont..... 28
Mill of..... 29
Silver slimes, analysis..... 40
Connell Ferry, Argyllshire..... 309
Consolidated Mining & Smelting Co., of Canada..... 135
Cook, C. H..... 458
Copper—
Alloyed with magnesium....475, 477
Deoxidizing..... 474
Flotation process.....41, 49, 78
Highland Valley mine..... 119
Indexing literature of..... 485
Partridge furnace for smelting.. 262
Potash from ores of..... 303
Refining..... 462
Core sand, *see* Foundry sand.
CORLESS, C. V.—
Paper by, on training for social responsibility.....491–506
Costs—
By-product coke..... 390
Concentration, vs. flotation...35, 36
Cyanidation, vs. flotation, Buffalo mine..... 12
Cyaniding, Ainsworth div., B.C. 116
Decantation..... 223
Electrolytic zinc..... 167
Iron and steel production.....
361–374, 385, 386
McIntyre mill, Porcupine..... 98
Silver concentrate, Cobalt dist., marketing..... 33
Silver precipitation..... 10
Smelting, Partridge furnace.... 264
Cottrell process—
Potash recovered by..... 303
" Flue dust..... 306
" Riverside, Cal..... 304
Countercurrent Decantation: EAMES
207–225
Cowles potash process..... 314
Crawford Bay, Kootenay Lake, B.C.—
Quartzite..... 233
Creosote—
Flotation process..... 39
" " tests.....42–50

PAGE

Crowsnest dist., B.C. and Alta.—
Coal mining methods....100–108

Crushing—
Buffalo mine, practice.... 13
Flotation, economic limit.... 30
Highland Valley mill.... 119
Kingdon lead mine.... 183
McIntyre mine, Porcupine.... 97
Molybdenite.... 124
Notes by Bell.... 108

Cuba—
Magnesite.... 239
Potash.... 301

Cushman and Coggershall potash process....316, 317

Cyanide—
Consumption, Buffalo mine.... 19
Strength of, for gold ores.... 214
Supply to Canada assured.... 1

Cyanide process—
Ainsworth div., B.C....111, 114, 116
Buffalo mine.... 19
Cobalt dist.... 11
Decantation.... 207
McIntyre mine, Porcupine....95–97

Cylinders—
Lubrication of.... 409

Cypress Hills, Sask.—
Clays of.... 229

D

Davidson, Dr. J. G.—
Notes by, on by-product coke.. 393

Davis, N. B.—
Notes by on Missanabi river clays.... 236
Paper by on refractory materials in Canada....226–236

Dawson, Y. T.... 197
Climate....197, 205

Decantation—
McIntyre mill, Porcupine.... 101
Notes by Eames....207–225

Denny, J. J.... 6, 7

Department of Mines, Canada—
Bituminous sands, report.... 447
Investigation of sand deposits.. 265
Molybdenite testing plant....
....125, 127, 129

Deseronto, Ont.—
Charcoal iron plant.... 380

Development of Canadian Magnesite: The: Roast....237–255

PAGE

Dewey system of indexing.481–483, 489

Dewey-Walter process.... 7

Dick, W. J.—
Discussion by on electro-chemical industries in Canada..468–470
Notes by on coal situation in Canada....443–446

Dickens Co., Texas—
Potash.... 311

Diedriche, Emil F.—
On compounding oils and greases 411

Dirt Hills, Sask.—
Clay.... 229

Distillation—
Oil from woods....52, 53

Distillery waste—
Potash from.... 302

Dolomite—
Grenville dist., Que....251, 252

Dominion Iron & Steel Co.—
Plant of....334, 335, 344
Products....358, 383, 384, 385
Notes by Jones.... 373

Dominion Reduction Co., Cobalt, Ont.... 28
Mill.... 29

Donald, J. R.—
Discussion by on Canadian magnesite.... 249

Dorfman, A.—
Paper by on milling practice at McIntyre mine....93–109

Dorr, J. V. N....208, 220

Dorr thickener.... 215

Drawings—
Filing and indexing.... 489

Dredging, *see* Gold dredging.

Dresser, John A.—
Discussion by on Canadian molybdenite.... 131
" iron and steel industries of Canada.... 378
" Quebec magnesite 252
Min. ref.... 506

Drummond, T. R.... 380

Drury, C. W.—
Paper by on potash, sources and uses....292–320

Drury feldspar-potash process.... 318

Drysdale, C. W.... 235

Duchy of Anhalt.... 294

PAGE

Durham, Edward B..... 482
Duriron..... 23
DYE, ROBERT E.—
Paper by on milling practice at the Buffalo mine.....11–27

E

EAMES, L. B.—
Paper by, on countercurrent decantation.....207–225
Eastern Car Co..... 335
Edmonton, Alta.—
Bituminous sand for street paving.....454, 456
Education—
Solution of industrial problems. 491–506
Educational reform—
Paper by Corless.....491–506
Electric smelting—
Iron..... 392
Electricity—
Canadian water-power, transmitted to U.S.....469, 470
Electro-chemical industries—
Canada.....460–470
" trade..... 469
Electro-deposition of Zinc from Aqueous Solutions: MATHEWSON... 135–160
Electro-metallurgical industries—
Canada.....460–470
Electro Zinc Co., Welland, Ont... 162
Electrolysis—
Magnesium..... 472
Zinc, paper by Mathewson.. 135–160
" " Watts.....161–167
Electrolytic copper—
Magnesium in producing...475, 480
Electrolytic Zinc in Eastern Canada: WATTS.....161–167
ELLS, S. C.—
Paper by on bituminous sands of northern Alberta.....447–459
Elmore vacuum process—
Molybdenite.....124, 126, 127
Epsom salts—
Manufacture in Canada..... 247
Explosives, *see also* Blasting.
Crowsnest dist., coal mines....406, 408

F

PAGE

Feldspar—
Potash from.....315–319
Felsite—
Coxheath, N.S..... 228
Ferro-alloys—
Canadian industry..... 468
Power for making..... 461
Ferro-molybdenum.....
122, 123, 126, 128, 134
Ferro-titanium..... 466
Fertilizer—
Potash for..... 292
Filing, *see* Indexing.
Filing cabinet—
Directions for making..... 484
Filtering—
Pulp, Buffalo mine..... 23
Fine grinding, *see* crushing.
Fire-brick—
Sydney mines..... 227
Fire clay—
Canadian occurrences.....227–230
Fish-water—
Potash from..... 302
Flotation—
Buffalo mines, paper by DYE.. 11–27
Chalcopyrite.....41, 49
Cobalt dist., oil used for..... 30
tests.....40–49
Paper by CALLOW and THORNHILL .28–37
Highland Valley mine.....119,120
Molybdenite.....121, 123–126, 132
Flotation concentrate—
Treatment of.....20, 33, 34
Flow sheet—
Buffalo mill.....14, 26
Cobalt dist. milling practice.... 29
Decantation..... 208
Flotation, Cobalt dist.....31, 32
Wood distillation.....52, 53
Flower Cove, N.B.
Refractory clay..... 228
Flue dust—
Potash from.....305, 306
Foreign Mines Development Co.. 311
Forest Products Laboratories—
Flotation oil experiments...1, 38–93
Formation of Zinc Ferrate, The: HAMILTON, MURRAY and MCINTOSH.....168–179
Foster, Rt. Hon. Sir George E.... 1, 4
Foundry pig iron, *see* Pig iron.

PAGE

Foundry sand—
Paper by COLE 265–291
France—
Coal trade 437, 438
Waterpowers 467
Frankenstein, Silesia—
Magnesite 238
Frechette, Howells—
Discussion by on calcining magnesite 248
Min. ref. 234, 249, 250
Freights, *see* Transportation.
Frenchman river, Sask. 229
Frick, F. F. 135
Frieslebenite—
Amenability to flotation 37
Furnace—
Linings for 245
Partridge, description and uses. 256–264
Further Notes on Yukon Mining Problems: PAYNE 188–206
Future of the Coal Trade of the Empire: GREENWELL 434–446

G

Galena, *see* lead.
Galetta, Fitzroy tp., Ont. 181
See also Kingdon lead mine.
Gas—
Blast-furnace, potash from 305
Gellivare, Sweden 398
General Naval Stoves Co., New York—
Oils distilled by 40, 43, 65
tests with 52
Geological Survey of Canada—
Magnesite discovered by 251
Geology—
Crowsnest dist., Alta 400
"German Kati Works" 299
Germany—
Coal trade 434, 435, 437, 438
Iron resources 397
Magnesite 238
Magnesium 471, 472
Potash, exports and resources ... 293
" industry 294–300
Gignon and Cagnage 309

PAGE

GILMORE, R. E.—
Paper by (and PARSONS) on oils for flotation 38-93
Gold—
Decantation process 210
McIntyre mine, Porcupine, Ont. 93, 94
Gold dredging—
Klondyke 195–197
Goldfield, Nevada—
Cyaniding practice 214
Gold milling—
Porcupine area, Ont. 93–109
GOODWIN, Dr. W. L.—
Discussion by on electro-industries 467
Granby smelter, Anyox, B.C.—
Coke for 390
Grand Lake coal area, N.B.—
Clay 228
Grant, E. P.—
Notes by on Molybdenite treatment at Renfrew and Orillia. 131–134
Gravel—
Thawing frozen 188–195
Great Britain—
Chemical products 469
Coal trade 434, 436–443
Iron and steel trade 397
Great Falls, Montana 136
Zinc refining 157, 160
Great Salt Lake, Utah—
Potash 311
Greece—
Magnesite 237–239, 242, 245, 247
GREEN, RAOUL—
Paper by, on mining methods in Crowsnest dist., B.C. 400–408
GREENWELL, ALLAN—
Paper on Empire's coal trade 434–446
Greenville area, Que.—
Magnesite 240, 251, 253–255
Greenville series—
Galena 181
Magnesite 234
" deposits 253–255
Grosvenor, Dr. Wm. H. 474
Gwillim, J. C.—
Discussion by on "electrolytic zinc" 157, 158

H PAGE

Haddow, A. W. 454
HAMILTON, E. H.—
Paper by, (and Murray and McIntosh) on formation of zinc ferrate 168–179
Hamilton, E. M. 6, 10
Hamilton, Ont.—
Iron industry 336, 337
Hansell, H. V. 391
HARDMAN, JOHN E.—
Paper by, on the Kingdon lead mine 180–187
Hardwood—
Distillation, flow sheet 53
For flotation 68, 74, 76, 79
Oil distillation from 39
Hargraves potash process 314
Hartsalz, *see* Potash—
Harz Mts.—
Potash 294
HASSELBRING, A.—
Paper by, on mining methods at Magpie iron mine 321–332
Headframe—
Magpie mine, photo 322
HEDLEY, ROBERT R.—
Paper on iron and steel industry for British Columbia 389–399
Helen iron mine, Michipicoten, Ont.—
Clay 228
Nearing exhaustion 380
Production 339
Hematite, *see* also Iron—
British Columbia occurrences .. 390
Highland Valley Mining and Development Co.—
Mine and mill of 118–120
Hodge, Dr. Edwin T.—
Discussion by, on British Columbia iron industry 398
Hoffman and Suchy 476
Hofman and Wells—
Zinc experiments of 169
Hollinger gold mine—
Cyaniding practice 218, 220, 221
Decantation plant 222–225
Holt-Dern roasting furnace 24
Notes by Callow and Thornhill 34
HOWE, H. E.—
Paper by, on electro-chemical industries in Canada 460–467

PAGE

Hudson Bay mine, Cobalt, Ont.—
Mill of 29
Hydro-magnesite, *see also* Magnesite—
Chemical formula 237

I

Imperial Munitions Board 126
Molybdenite supplied to ... 126, 134
India—
Coal trade 434, 438
Magnesite 238
Potash 295, 301
Industrial problem—
Solution of 491–506
Inland Crystal Salt Co 312
Intercolonial Coal Co., Westville, N.S 227
Interior Plateau, B.C.—
Clays 230
International Molybdenum Co.
Ferro-molybdenum produced by 128
Works of 121, 127
Notes by Grant 131–134
Inyo Development Co 311
Iodine—
From kelp 308, 309
Iron—
British Columbia industry .. 389–399
Canadian industry 333–388
Elimination of from zinc 167–179
Magpie mine, methods of working 321–332
Iroquois lake—
Moulding sand 267
IRWIN, G. A.—
Discussion by, on Canadian iron and steel industries 374
Italy—
Alunite 306
Magnesite 238

J

Janney testing machine—
Description 41
Japan—
Coal trade 434, 437, 438
Potash industry 309
JEVONS, STANLEY—
Quoted on "The Coal Question" 439, 440

PAGE

Jones, C. C. 396
JONES, F. P.—
Discussion by, on position of Canadian iron and steel industries 373, 374
Jones, Thos. R. 11, 78
Jordon, Fred. A 390

K

Kainite, *see* Potash.
Kalusz, Galicia 295–300
Kaolin—
British Columbia 230
St. Remi, Que 228
Keeney, Robert M. 481, 489
KEFFER, FREDERIC—
Paper by, on Highland Valley mine and mill 118–120
Kelp—
Potash from 308
Kenner, Alvin R. 482
Ketone oils—
Distillation 69
Flotation process 39, 73, 77
" tests 46, 60, 61
Khewra, India 301
Kieserit 295
Killarney, Ont.—
Quartzite 232, 233
KING, J. C.—
Paper by, on Magnesium ... 471–480
Kingdon Lead Mine, The: HARDMAN 180–187
Kinnaird St., Edmonton 454
Kirkpatrick, Prof. S. F 167
Kiruna, Sweden 398
Klondyke, Y.T.—
Climate and precipitation ... 198–205
Gold dredging 195–197
Klugh, B. G. 390
Kootenay Districts, B.C.—
Ore-dressing plants 117
Kootenay lake, B.C.—
Quartzites 233
Kyuquot, B.C.—
Fireclay 230

L

Labour—
Hollinger decantation plant 223
Strikes, cause of 494

PAGE

Laflamme Id., *see* Chat Island.
Laist, Frederick 135, 136
Lake Agassiz, Man.—
Moulding sand 267
Lake-of-the-Rivers, Sask 229
La Riviere, Man. 229
LATHE, FRANK E.—
Paper by, on indexing technical literature 481–490
Lawson Bros., Ottawa—
Moulding sands tested by 278, 286, 287, 288
Leaching gold ores—
McIntyre mine 95, 96
Lead—
Alloyed with cadmium and magnesium 478
Eastern Ontario 180
See also Kingdon mine.
Indexing literature of 485
Partridge furnace for smelting. 262, 264
Refining 462
Le Blanc soda process 309
Leucite—
Potash from 314
Leucite Hills, Wyoming 315
Leopoldshall, Germany 294
Levant tp., Ont.—
Quartz 232
Lena River goldfields, Siberia 197
Lignite, *see also* Coal.—
Pulverization for locomotive fuel 424–430
Lillooet mining div., B.C.—
Chromite 231
Magnesite 235
Lime—
Cyaniding, Buffalo mine, consumption 20
Lindeman, R. 394
Little Alkali lake, Ore 312
Locomotive engines—
Pulverized fuel for 416–433
Logan, Sir W. 180, 181
Longford Mills, Ont.—
Oil distillery 39, 40
Lorraine, Germany—
Iron mines 397
Lowell, Mass. 461

PAGE

Lubricating oil—
Consumption record 415
Filtration of 412
Storage 414
Lubrication of Mine Machinery: BOYD 409–415

M

McAllister, W. B 250
McAllister & Boshart 250
MacDowell, Charles H 307
Macedonia—
Magnesite 238
Machinery—
Lubrication of 409–415
McINTOSH, D.—
Paper by, (and Hamilton and Murray) on formation of zinc ferrate 168–179
McIntyre gold mine, Porcupine, Ont.—
Milling practice 93–109
Mackenzie, G. C.—
Acknowledgements to 129
Min. ref 78
Mackinaw—
Sketch of 330, 331
McKinley-Darragh-Savage Mines, Cobalt, Ont.
Flotation plant 28
" results 30, 33
Mill 29
McKune, F. R 245
McMurray Alberta—
Bituminous sand ... 452, 454, 457, 458
Magdalena, Mexico—
Lead-silver smelting 264
Magnalium 480
Magnesite:
Calcining 248–250
Canadian industry 237–255
" occurrences 233–235
Magnesium mfg 473
Magnesite brick 244
Magnesium—
Manufacture of in Canada 245
Magnesium chlorine 473
Magnesium: Its Properties, Manufacture and Uses: KING 471–480
Magnetite, *see also* Iron—
British Columbia 390
Magpie iron mine 321
Sudbury dist 380

PAGE

Magpie iron mine, Michipicoten dist., Ont.—
Mining methods at 321–332
Production of 339
Manchester, N.H 461
Manganese—
Alloyed with magnesium 480
Magpie iron mine 321
Manganese bronze 480
Manitoba—
Fuel and power problems 445
Refractories, imported 227
Refractory clays 229
Manure salts, *see* Potash.—
Margarita, Mexico—
Magnesite 239
Marsac mill, Park City, Utah 7
Marysville, Utah—
Alunite 306
MATHEWSON, E. P.—
Discussion by on "Electrolytic Zinc" 157–160
Min. ref 157
Paper by, on "Electrolytic Zinc" 135–157
Matildite—
Amenability to flotation 37
Mellon Institute 447
Metallurgical Section of Institute. 2
Metallurgy—
Indexing literature of 484
Metals—
High price of 1
Methods of Mining in Crowsnest Pass District, Alta.: GREEN 400–408
Mesabi iron ore—
Price of 362
Michipicoten dist., Ont.—
Iron mining, *see* Magpie mine.
Militia Department—
Institute co-operating with 1
Milling, *see* Gold milling, amalgamation, etc.
Milling Practice at the Buffalo Mines, Cobalt, Ont.: DYE 11–27
Minas Geraes, Brazil 398
Mine cars—
Lubrication of 413
Mineral production—
Magnesite, Canada 234
Molybdenum 127

PAGE

Minerals Products Corporation—
Potash process 307
" production 306
Mines Branch, Ottawa—
See also Canadian Department of Mines.
Flotation oils, experiments of... ..... 1, 38-93
Mining companies—
Libraries of, indexing.....481–490
Mining methods—
Crowsnest dist., Alta..... 401
Mining Methods at the Magpie Iron Mine: Hasselbring.....321–332
Missanaibi river, Ont.—
Clays.....228, 235, 236
Fireclay..... 228
Mitchellton, Sask. —
Clay near..... 229
Mocorito, Sinaloa, Mexico—
Decantation..... 208
Molasses—
Potash from..... 303
Molybdenite—
Concentration and marketing.. 121–134
Molybdenum Mining & Reduction Co..... 129
Molybdic acid..... 134
Montauban tp., Portneuf Co., Que.—
Zinc mining 161
Montreal, Que.—
Refractory ware..... 227
Moose Jaw, Sask.—
Clay works near..... 229
Moose Mt., Ont.—
Iron ores..... 379
treatment of...380, 390, 391
Morrow, J. G.—
Discussion by on Canadian iron and steel industries.....377, 378
Moulding sand—
See also Foundry sand.
Occurrence and testing.....265–291
Photomicros.....281–283
Mount St. Patrick, Ont.—
Molybdenite..... 126
Muriate of potash.....295, 296, 298
Murray, G.—
Paper by, (and Hamilton and McIntosh) on formation of zinc ferrate.....168–179

PAGE

Musquodoboit, N.S.—
Clay..... 228

N

Nails, *see* Wire Nails—
National mines, Cobalt, Ont.... 28
Mill..... 29
Nebraska—
Potash.....310, 312
Nevada—
Alunite..... 306
Potash..... 313
New Brunswick—
Coal, analysis..... 425
Refractory clay..... 228
Newfoundland—
Coal imports.....435, 436
Newnan, W. E..... 184
Newnan furnace..... 184
New York State—
Power, market value..... 470
Niagara Falls, Ont. and N.Y.—
Waterpower..... 461
exported to U. S..... 470
Nickel—
Refining..... 461
Nicola dist., B.C.—
Coking coal.....394, 395
Nipissing silver mine, Cobalt, Ont.—
Flotation plant..... 28
Mill..... 29
Sodium sulphide precipitation..6–10
Nitrogen—
Taxation of.....461, 466
Non-Bessemer iron ore—
Price of..... 362
North American Magnesite Co... 248
Northern Customs Concentrators, Cobalt, Ont..... 28
Mill of..... 29
Norway—
Electric power, cost..... 462
Electro-chemical industries.... 469
Magnesite..... 238
Molybdenite mining.....124, 126
Notes concerning Local Treatment for certain West Kootenay Silver Ores: Thompson.....110–117
Notes on Electro-Chemical and Metallurgical Possibilities in Canada: Howe.....460–467
Discussion.....467–470

PAGE

Notes on the Highland Valley Mine and mill near Ashcroft, B.C.: KEFFER 118–120
Notes on Milling Practice at the McIntyre mine, Porcupine, Ont.: DORFMAN 93–109
Notre Dame des Anges, Que.—
Zinc refining 165, 167
Nova Scotia—
Coal, analysis 425
" reserves 432, 436
Iron and steel industries 334, 335, 344, 386, 387
Refractories imported 226
Refractory clay 227, 228
Nova Scotia Steel and Coal Co.—
Fire-brick plant 227
Plant of 335, 344, 350
Products 349
Notes by Jones 373
Nurpur, India 301

O

O'Brien mine, Cobalt, Ont —
Mill 29
Occurrence and Testing of Foundry Moulding Sands, The: COLE, L. H. 265–291
Oil—
Flotation process 120
" Cobalt dist 30
" " experiments 38–93
Fractional distillation 54
From bituminous sand 459
Oil flotation, *see* Flotation process—
Old Range iron ore—
Price of 362
Ontario—
Iron and steel industries 344
Refractory clay 228
Opportunities for the Establishment of an Iron and Steel Industry in British Columbia: HEDLEY . . 389–399
Oregon—
Potash 310, 312
Orikany sandstone 233
Orillia, Ont —
Ferro-molybdenum 126, 127
Molybdenum refinery 128, 133
Owen's lake, Cal.—
Potash 311
Oxygen—
Affinity of magnesium for 475

P

PAGE

Page-Hersey Iron Tube and Lead Company 355
PARSONS, C S.—
Paper by, (and Gilmore) on oils for flotation 38–93
PARTRIDGE, ALLEN R.—
Paper by, on Partridge furnace 256–264
Partridge Smelting Furnace, The: PARTRIDGE 256–264
Paving—
Bituminous sand for 452
Edmonton 454
United States 456
PAYNE, HENRY M.—
Paper by, on Yukon mining problems 188–206
Peat—
As a locomotive fuel 433
Development of Canadian industry 445
Penn Canadian mine, Cobalt, Ont.—
Mill of 29
Peridotite—
Northern Ontario, nickeliferrous ores 236
Peru—
Potash 301
Phonolite—
Potash from 315
Phosphor bronze 480
Pig iron, *see also* Iron—
Canadian consumption 373, 382
" production 345, 382
" statistics 342, 343
Costs of producing 362, 363, 365, 366, 392, 393
United States 382, 397
Pine—
Oil from, for flotation 40, 57–64
Tests 42, 43
Pine oil, *see also* Oil—
Flotation process 38, 57
Frothing 64
Pistor and Rakowics 476
Plans—
Magpie iron mine 328, 329
Platinum—
Combined with magnesium, effect 472
Porcupine area, Ont.—
Cyaniding practice 214, 217, 219
Decantation 222–225
Milling practice 93–109

PAGE

Porter, Dr. J. B. 378, 506
Potash, its Present and other Possible Sources: DRURY 292–320
"Potash Syndicate"—
History of 296
Potsdam sandstone—
For glass making 233
Potter Co., Texas—
Potash 311
Power, *see also* Water-power—
Dredging, Yukon 196
Electro-chemical industries 460–464, 467
Hollinger decantation plant 223
Hydro-electric, notes by Howe. 460
Canada, exported to U.S. 469
Precipitation—
Klondyke and Siberia 203–205
Present Flotation Practice on Cobalt Ores: CALLOW and THORNHILL. 28–37
Present Position and Future of the Iron and Steel Industries of Canada 333–372
Discussion 372–388
Presidential Address 1–5
Price, J. R 458
Production, *see also* Statistics—
Asphalt, United States 448
Coal, notes by Greenwell ... 434–446
Pig iron, Canada 345
" " United States 397
Steel products 345, 346, 348, 360
Prohibition 50
Proustite—
Amenability to flotation 36, 37
Pulverized Fuel for Locomotives: COFFIN 416–433
Pyrargyrite—
Amenability to flotation 36, 37
Pyrite—
McIntyre mine, Porcupine 100

Q

Quartz—
For silica brick 232
McIntyre mine, time of settling 102, 103
Quartzite—
Canada 232, 233
For silica brick 231, 232

PAGE

Quebec—
Chromite 230
Magnesite 249
" notes by Davis 234
" origin 254
" paper by Roast ... 237-248
Refractories, imported 226
Refractory clay 228
Titaniferous iron 379
Zinc mining 161
Quyon, Que.—
Molybdenite 126

R

Railroad Valley Co., Nevada ... 313
Railways—
Electrification of, in Canada ... 445
Randall, John—
Decantation process of 207
Randall Co., Texas—
Potash 311
Refining—
Metals, power required 461, 462
Refractories, *see also* Magnesite
Refractory brick—
Magnesite 245
Refractory Materials in Canada: DAVIS 226–236
Reid, Fraser D 28, 29
Renfrew, Ont.—
Molybdenite concentrator 121, 127, 132
Renfrew Molybdenum Mines . 126, 127
Reports—
Filing and indexing 481
Resinous wood—
Oils from 65, 66
Rhodochrosite—
Magpie iron mine 321
Richards, Prof. J. W 473
Riverside Portland Cement Co. 303, 304
ROAST, HAROLD J.
Discussion by on "Electrolytic Zinc" 159
Paper by, on Canadian magnesite industry 237–248
Views on calcining magnesite .. 250
Roasting—
Iron, Magpie mine 321, 322, 331
" photo of plant 323, 324, 325

PAGE

Roasting furnace—
Flotation concentrate, Buffalo mine 22
Robbins, P. A 225
Robertson, James 180
Roche, Hon. W. J. 78
Roscoe and Schorlemmer 168
Ross, H. J. 248
Rubel, Walter 477
Russia—
Coal trade 437–439
Magnesite 238
Potash 295, 302

S

St. John, Que.—
Refractory ware 227
St. Lawrence river—
Moulding sand 267
St. Remi, Que.—
Kaolin 228
Quartzite 232, 233
Salduro salt marsh, Utah 313
Sand, *see* Foundry sand—
San Pedro, Cal.—
Potash works 311
Saskatchewan—
Coal, analysis 425
" reserves 432
Firebrick 227
Refractories, imported 227
Refractory clay 229
Sawdust—
Potash from 302
Scales, automatic—
Magpie iron mine, photo 326
Schist—
McIntyre mill, time of settling 102, 103
Scotland—
Kelp industry 309 310
Scottie Creek, B.C. 231
Searles lake, Cal. 310
Security Cement and Lime Co ... 303
Settling—
McIntyre mill, Porcupine 101
Shaft—
Magpie mine 322, 323
Shawinigan Electro Metals Co.—
Magnesium produced by 473, 477

PAGE

Shawinigan Falls, Que.
Electric zinc furnace 162
Shipbuilding—
British Columbia 396
Government assistance to 376
Shubenacadie, N.S.—
Refractory clay 227
Siberia—
Climate and precipitation .. 197–205
Siderite, *see also* Iron
Magpie mine 321
Michipicoten, Ont 379
Silica—
Refractory clay 231
Silica brick 231
Silicate rocks—
Potash from 313
Silver—
Kingdon lead mine 182
Market conditions, 1916 1
Refining 24
Sodium sulphide precipitation .. 6–10
Silver Glance mine, near Kaslo, B.C. 110
Silver Hoard mine, B.C.
Tests on ore from 112
Silver (metallic)—
Amenability to flotation 37
Sisquoc, Cal.—
Bituminous sand, plant 450
Skip—
Magpie iron mine 327
Smelting—
Lead, Kingdon mine 185
Molybdenite 122
Partridge furnace for 256–264
Smith, C. Dupre 208
Social problems—
Notes by Corless 491–506
Sodium sulphide—
Silver precipitant 6, 24
" cost 10
Sodium Sulphide Precipitation of Silver at the Nipissing Mine: WATSON 6–10
Some Suggestions on Filing and Indexing: LATHE 481–490
South Africa, *see* Africa, S.
South America, *see* America, South.
Spain—
Alunite 306
Coal trade 438, 439
Potash 301

PAGE

Standard Chemical Co., Longford Mills 39
Oils of 41, 42
" tests for flotation 52
Standard Chemical, Iron and Lumber Co. 53
Standard Iron Co., Deseronto, Ont. 344, 380
Stansfield, Dr. Alfred—
Electric furnace designed by ... 162
Min. ref 167, 248
Stassfurt, Germany—
Potash 293, 294, 295
Stationary boilers, *see* Boilers
Statistics, *see also* Mineral Production and Production
Coal, Empire's trade ... 434–438, 440
Iron and steel, Canada 342, 349–351, ... 353, 356, 360, 373, 383–385
Steam cylinder, *see* Cylinders
Steel—
British Columbia, prospects for industry 389–399
Canadian industry 333–338
Steel Company of Canada—
Plant of 336, 338, 344, 352, 385
" well situated 377
Products 355, 358, 383
Notes by Jones 373
Steel rails—
Production, notes by Whitton 346, 387, 388
Stege, California 263
Stewart, R. H.—
Discussion by on "electrolytic zinc" 157, 158
Stirling boiler—
Pulverized fuel used with 430
Streets—
Bituminous sand for paving .452, 456
Sudbury dist., Ont.—
Flotation process 78
Iron ore, west of 380
Sugar—
Potash from residues 302
Sullivan mine, Kimberly, B.C.—
Zinc ore, analysis 162
" " assays 172, 173
" " microscopic examination 163
" " roasting tests 173–179
Sulphate of magnesium, *see* Epsom salts

PAGE

Sulphate of potash 295, 296
Sumas, Mt., B.C.—
Clays 230
Sumner lake, Ore 312
Summit series, Kootenay dist., B.C. 233
Sweden:
Coal trade 438
Pulverized fuel on railways in .. 417
Sydney, B.C.—
Potash works 310
Syvinit, *see* potash.

T

Tailing—
Cobalt dist., treatment of 37
Tantallon, Man. 229
Tariff—
Steel and steel products, ... 353, 354, 357, 358, 375–377
Tassoo Harbour, Queen Charlotte Islands—
Copper ore 391
Taylor creek, Lilloett dist., B.C. ... 231
Technical literature—
Indexing and filing 481–490
Temiskaming mine, Cobalt, Ont.—
Mill of 29
Temperatures—
Klondyke and Siberia .. 198–202, 205
Tepic, Mexico—
Smelting 263, 264
Tetreault zinc mine, Que. 161
Texas—
Potash 310, 311
Thallium—
Alloyed with magnesium 478, 479, 480
Thawing—
Yukon, frozen gravels 188–195
Thompson, Nichol—
Discussion by on British Columbia iron and steel industry ... 394
Thomson, Francis A.—
Paper by on treatment of certain Kootenay ores 110–117
Thornhill, E. B.—
Min. ref 24
Paper by, (and Callow) on milling practice in Cobalt dist .. 28–37

PAGE

Timbering—
Crowsnest dist., coal mines....402, 403
Tin—
Alloyed with magnesium..... 478
Tin plate..... 355
Tintic Milling Co., Silver City, Utah..... 35
Titaniferous iron—
Quebec resources..... 379
Titanium—
Alloyed with magnesium..... 479
Touceda, E. A.....478, 479
Trail, B.C.—
Electrolytic zinc refining..... 157
Transportation—
Bituminous sand..... 457
Canadian iron and steel industry, costs of.....364, 371, 372, 386
Coal, in Canada..... 435
Transvaal—
Magnesite..... 238
Trethewey mine, Cobalt, Ont.—
Mill of..... 29
Trona, Cal., *see* Searles lake
Tulameen, B.C.—
Chromite..... 231
Turnbull, John M.—
Discussion by on British Columbia iron industry..... 397
Turtle Mt., Man..... 229
Turpentine—
Oil flotation.....59, 62
Twelve Mile lake, Sask..... 229
Two Mountains, Que.—
Clay..... 228
Tyrrell, J. B.—
Discussion by, on Canadian molybdenite..... 129
Min. ref..... 391

U

United Kingdom, *see* Great Britain.
United States—
Bituminous sand deposits..... 448
Coal trade...434, 437, 438, 441–444
Electro-chemical industries.... 469
Iron industry.....
340, 341, 346, 347, 361, 397
Population and industries..... 494

United States—Con. PAGE
Potash.....310–313
" investigation..... 300
Railways, pulverized coal used in 417
United States Smelting and Refining Co..... 306
United States Steel Corporation—
Notes on, by Whitton..340, 361, 363
Ref.....374, 397
Universal Portland Cement Co... 303
Utah—
Potash.....311, 313
Utah Chemical Co..... 311
Utah Potash Co—
Potash process..... 307
Ref..... 306

V

Vancouver Gas Co..... 394
Vancouver Island, B.C.—
Alunite..... 306
Coal, analysis..... 425
Coking coal..... 394
Venezuela—
Magnesite..... 239
Ventilation—
Coal mines.....406, 408
Verona, Ont.—
Quartz..... 232
Vickers, Chas..... 477
Virgin River salt deposits, Nevada 313
Vocational training—
Notes by Corless.....494, 501
Vulture Mines Co., Wickenburg, Arizona..... 208

W

Wabana, Newfoundland—
Iron deposits..... 398
War, The Great—
Effect of, in Canada.....186, 187
Empire's coal trade affected by 434–446
Industrial problems.....491, 497
Iron resources in relation to.... 397
Magnesium, Canadian industry 471
" uses..... 473
Water-power—
Electro-chemical industries.460–470

PAGE

WATSON, R. B.—
Paper by, on sodium sulphide precipitation of silver 6–10
WATTS, E. E.—
Discussion by, on "electrolytic zinc" 158
Paper by, on electrolytic zinc in eastern Canada 161–167
Watts electrolytic process 164
Wealth—
Distribution of 503, 504
Weedon Mining Co. 165
Welland, Ont.—
Zinc refining 162, 165
Westville, N.S.—
Refractory clay 227
White, A. V. 470
WHITTON, CORBETT F.—
Paper on iron and steel industries of Canada 333–372
Discussion 381–388
Ref. to 373
Willows, Sask 229
Wilson, M. E.—
Discussion by, on Quebec magnesite deposits 250, 252
Wire nails—
Canadian production 360
Freight rates on 372
Wood—
Distillation of 52, 53
Potash from 302

PAGE

Wood ash—
Potash from 301
Wood flotation machine 125
Wood, H. E.—
Min. ref. 121
Molybdenum testing plant 127
Wood pulp—
Magnesite used in making 244
Wyoming—
Leucite 314
Wyomingite—
Potash from 315

Y

Yukon—
Mining problems, paper by Payne 188–206

Z

Zinc—
Alloyed with magnesium . . . 477, 480
Electrolysis 168
" paper by MATHEWSON 135-160
" paper by WATTS . 161-167
Zinc Commission 163
Zinc Company Limited—
Plant of 161
Zinc ferrate—
Formation of 168–179
Properties of 171

BINDING SECT. JUL 19 1978

TN
1
C2
v.20

Canadian Institute of
Mining and Metallurgy
Transactions

19

Engineering

PLEASE DO NOT REMOVE
CARDS OR SLIPS FROM THIS POCKET

UNIVERSITY OF TORONTO LIBRARY

ENGIN STORAGE